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J. W. RICHARDS, PH. D. ..... *President.*
E. F. ROEBER, PH. D. ..... *Editor.*

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ADVERTISING RATES GIVEN ON APPLICATION.

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Boston Meeting American Electrochemical Society.

The programme published on another page of this issue for the annual meeting of the American Electrochemical Society, to be held during the last week of this month in Boston, shows that the local Committee, together with the Secretary of the Society have done energetic and excellent work toward making this meeting an especially memorable one. The list of papers, as far as it is published, is well balanced between scientific research and engineering application. The same fundamental tendency towards harmony between the art and the science is happily indicated by the cordial way in which both Harvard University and the Massachusetts Institute of Technology will extend alternately their hospitality to the Society. We are informed that a comparatively large number of the members of the society have already declared their intention to attend the meeting.

Phases of Development in a New Process.

In the constant state of flux of metallurgy, numerous attempts to improve the present practice in details, or to develop some radically new process, are always being made. Growth is a necessary concomitant to life. A concern that does not grow is liable to decay. Consequently, an attitude of always trying to evolve something better from the old is a healthy one. Eternal experimenting is the price of progress. The course of development of a new process is susceptible to a searching out of the essential qualities for success.

Every successful process must be first of all scientifically possible. It is impossible to make water run up hill. It is likewise impossible to make a reaction proceed when the free energy of the counter reaction is against it. Next, the apparatus must be mechanically satisfactory. A furnace designed without allowing for the expansion of the firebricks, or a vat that is slowly corroded by the liquor, will cause the practical failure of the most beautiful set of reactions. Finally, and most important of all, the process should work on such raw materials and should produce such a product that it can be sold at a profit under market conditions. The ability to return profit to capital is the last test of any process. Very few of countless new ideas ever are commercially successful. The reason for this lies in the fact that all three factors must be present. Suppose all but one out of ten new proposals fail for scientific weakness. And suppose that all but one of the survivors fail for engineering weakness. Finally, suppose that all but one of a hundred fail for commercial reasons. On these assumptions, out of ten thousand new ideas only one has real merit. Many of these that possess real merit are never commercially successful because of poor handling, like "the seeds that fall by the wayside." The small number of

successful new projects is always characteristic of progress. As Michael Faraday says: "The world little knows how many of the theories of the scientific investigator have been crushed in silence by his own severe criticism; that in the most successful instances not a tenth of the hopes, wishes, the preliminary conclusions have been realized." Of the million of fish eggs of a shad, but few reach maturity.

Such being the case concerning a new process, a course of action in developing it should be followed that corresponds to these conditions, and in the order named. The first test should be to see if the process is scientifically all right. This can be done on a small scale, and the expense of such a test is slight. It should be next tested to see if the conditions found right in small experiments can be held on a larger scale. The final test is how efficiently the process works on a very large scale, and thus it can be estimated whether it will be a success. Such being the plan of campaign, it should logically and methodically be carried out. Every step should be the result of frank discussion by those interested. Now, the new process will be found to advance by the following stages, something according to the general scheme outlined above. First, the laboratory stage: Here it must be shown that it is scientifically possible to produce the results on a small scale. Second, the practical stage: This should be on a fairly large scale, say, 300 pounds of the material, and show that it is possible to hold the right conditions, found in the laboratory stage, for some time. Third, the engineering stage: This should operate on one or two tons of the material continuously for a long period. Many defects must be shown up and remedied. The apparatus must be proved durable and efficient, and the recovery must be high. The control over the purity of product must be shown. In brief, this third stage must give evidence that the passage to the fourth one is certain. This fourth stage is the commercial stage: Actual production of finished material for the market at a greater profit than under the old process. This is the goal toward which all the energies had been turned—commercial success.

From each stage it must be demonstrated that progress to next stage is as sure as things can be in this world of uncertainty. By such a careful, well-thought-out, conservative course, can the risk incident to starting a new enterprise be reduced to a minimum. In war, it is said that one ton of lead is necessary to kill each soldier. So many shots miss the mark. The same is true of industrial work. Well-planned schemes, carried out with reason and courage, will win in the end, barring chance and the happening of the unexpected. But in metallurgy, as in the whole world, nothing succeeds like success.

Thermochemical Data for Metallurgists.

Numerous letters which have reached this journal, make it evident that the serial on metallurgical calculations from the pen of Prof. J. W. Richards, which was commenced in our last issue, is being received very cordially and even with enthusiasm by progressive practical metallurgists. The second installment, which is given in the present issue, is essentially a summary of those thermochemical data which should be of most importance for metallurgists. The thermochemical balance sheet of a metallurgical process should have the same importance for the engineer in charge as the financial balance sheet has for the business manager. It shows the efficiency of the whole process and its various parts; it makes it clear where it would be wise to try and make improvements, and where the highest possible efficiency has already been attained and further attempts of improvements must be failures. Prof. Richards' careful compilation and arrangement of those most important thermochemical data which have been determined, should, therefore, be much appreciated, since very few metallurgists have at their disposal the voluminous works of Berthelot and Thomsen. But, alas! how few are the data in some fields where the metallurgist needs them so badly—for instance, data on silicates and alloys. True, every scientific research, if made in the right spirit, will yield fruitful results. Nevertheless, it would seem that much time, money and effort is now spent on so many researches, the fruitfulness of which is not quite evident (as in organic chemistry), while no one knows the heat of formation of blast furnace slag, or the specific heat of iron ore, Fe_2O_3, to the temperature of the blast furnace, or the heat of formation of the most important alloys. Funds, granted by the Carnegie Institution for the determination of these badly needed data, would be well spent.

Large Scale Production and American Metallurgical Practice.

If there is any difference at all between American and European metallurgy, such difference lies in the magnitude of the operations in America. In what is familiarly termed "the steel business," by those frank, energetic, breezy individuals "the steel men," we have seen for the past one hundred years a gradual increase in size of the furnace. The first small anthracite stacks of the Lehigh Valley and the wee little charcoal cupolas of the revolutionary time would not make each as much iron in a month as one of the monster 700-ton blast furnaces of the Carnegie Steel Co. or the Lackawanna Steel Co. do in a day. In lead smelting, there has been a constant increase to 200 tons capacity. Here there has been a halt, for lead smelting is a delicate operation, and too large units are unwieldy. But the Pueblo furnaces are large in comparison to the German furnace. Twenty-five years ago the usual capacity of the copper cupola was 50 tons a day, and the furnace was tapped intermittently. Mr. John L. Thompson, then superintendent of the Orford Copper Co., built the brick "raschette" furnaces of capacity of one hundred and fifty tons with the continuous siphon spout. This furnace was the subject of much discussion, and paved the way for the modern large water-jacketed blast furnace of 500 tons, such as those used at Ducktown and at Granby. To-day these are surpassed by the long furnaces of the Anaconda Copper Co. of 1200 tons capacity. In the reverberatory refining furnace in 1890 15 tons was a large capacity. The Baltimore Copper Co. soon increased this to 35 tons. Later, the Orford Copper Co. built its 50-ton furnaces. About four or five years ago the

Guggenheims built, at Perth Amboy, furnaces of 100-tons capacity, and later of even 150-tons capacity. These furnaces were built on a new design, and such a size as was only rendered possible by the use of the Walker casting machines.

This brief recital of individual achievements in certain lines is but typical of the general expansion of the metallurgical industry in the United States. The causes for this large development can be found in the peculiar conditions of American industry. In the first place there has always been an abundance of raw material. It is obvious that to handle large amounts of raw material, large units are necessary. Large reserves of raw material justify the erection of considerable machinery. In the next place unskilled labor is high, and it is inefficient. Skilled labor is still higher, but it is efficient. Consider the case of the large Anaconda copper furnace making 700 tons of slag a day. Careful work will keep this slag clean below 0.50 per cent Cu. Poor work can make a foul slag as high as 1.00 per cent Cu. This difference, 0.50 per cent Cu, or 10 pounds of copper to the ton worth, say, 10 cents a pound "in the matte," will amount to a dollar a ton or $700 total in a day. As such a furnace will employ eighteen to twenty men a day, at a total labor charge of not over $70, it is plain the price paid labor is low, provided it is efficient. Poor furnace work would be dear at any price, however low. Thus, the high price of American labor forces large installations, and these installations give the high-priced workman a chance to handle many tons of stuff efficiently, and so to work to advantage. The result of it all is that labor charge per ton is usually smaller in America than in Europe.

While labor conditions and cheap raw material have been the main causes, there are other contributing factors. One of these is that thermal efficiency is increased with increased size, just as is the case with electric furnaces. Thus, amount of fuel per ton of material treated lessens with increase of size. For instance, in the old copper-refining furnace the coal used was 600 pounds to the ton of copper. This was reduced successively to 400-300 down to 200 pounds in the 50-ton furnace, and, finally, in the case of the 150-ton furnace, to as low as 100-125 pounds. Although coal is usually cheap in this country, yet distance from the mine makes it often expensive and the cost, whatever it is, is cut down greatly. Again, one large furnace is less in capital cost than ten small furnaces each of one-tenth the capacity. Thus, in a large scale plant less money is tied up in furnaces, and more capital can be expended for auxiliary machinery. All these different causes have interacted and changed the complexion of the whole situation. They have also acted quickly, because the temperament of the American is conducive to change, especially if the change is one from smallness to bigness and cost of change will be soon paid for by cheaper work. The spirit of the American pioneers is seen to-day in American business enterprise. But we believe that in many branches of metallurgy the limit in size has been about reached. Increase will be made, but these will be in special lines. Further progress will be made in bringing all the plants of the country to more or less standard size. The real field of the future lies in the refinements of the apparatus, in the manipulation of the process, and in the development of by-products. In this Germany has been in the lead. But in Europe they are building large American blast furnaces. So America will develop the scientific side of metallurgy. Such a rivalry has more reason for the expenditure of productive capital than has the emulation of each other in building expensive destructive battleships.

Electro-Cyanide Process.

A patent recently granted to Mr. E. L. Oliver and abstracted in the present issue, is interesting in various respects, although nothing is known concerning the operation of the process on a commercial scale. While in all commercially successful processes using electrolytic precipitation of gold from cyanide solutions, the solution of the gold is carried on in leaching tanks and the solution is afterwards withdrawn and the gold is precipitated from it in other tanks, the endeavor of numerous inventors during recent years has been to carry on both solution and precipitation simultaneously in the same tanks. The process of Mr. Oliver belongs to this class. While the advantages of such a process would be evident, yet, as a matter of fact, none of the processes of this kind has so far worked really well on a large scale, although several of them appeared quite promising on a laboratory scale. Dr. W. H. Walker, in discussing this subject two years ago in an American Electrochemical Society paper, pointed out what may have been some reasons of such failures. The fact that tellurides of gold are insoluble in potassium cyanide, and so far as electrolytic methods are concerned can be reduced only by direct cathodic reduction, explains some of the earlier failures, but would also seem to show the way towards improvements. If cathodic reduction of the ore particles is to be effected, it is necessary to bring every minute particle of ore in effective contact with the cathode.

While nothing is said of cathodic reduction in the patent under discussion, it would seem that Mr. Oliver has been working along these lines of thought. As his cell is constructed, the ore particles are carried along the amalgamated cathode plates, and will make effective contact with the same, and the inventor also claims to have escaped what would be a very great drawback to his scheme—namely, the scouring of the amalgamated plates by the ore particles passing along their surface. The method by which circulation is effected is also interesting, although not new; it depends on the fact that if in two communicating tubes containing the same liquid, a gas is supplied to the liquid in one tube, the latter will become lighter and will rise and may be caused to overflow into the other tube. This continuous circulation may be effected, and, by varying the gas supply, the circulation can be regulated. This same principle of agitation is successfully used in another electrochemical apparatus where thorough stirring is of decisive importnace; namely, in the hypochlorite cell of Haas and Oettel; in the latter case no gas or air is supplied from an outside source, but the hydrogen which is developed at the cathode is directly made use of.

American Electrochemical Society.

The annual meeting of the American Electrochemical Society will be held in Boston on April 25 to 27. Professor W. H. Walker, of the Massachusetts Institute of Technology is chairman of the local committee. The hotel headquarters will be at the Hotel Lenox, Boyleston & Exeter Streets, Boston. The programme is as follows:

TUESDAY.

9.30 A. M. Session of the Society in Room 6 Lowell Building, Massachusetts Institute of Technology.
1 P. M. Reception by President Pritchett at the Technology Club, followed by lunch tendered by the Institute of Technology.
2 P. M. Excursion to the General Electric Company's plant at Lynn.
8 P. M. Presidential Address at 6 Lowell Building, followed by a smoker at Hotel Lenox.

WEDNESDAY.

9.30 A. M. Session of the Society at Harvard University, Peirce Hall.
1 P. M. Reception by President Eliot at Harvard Union, followed by lunch tendered by Harvard University.
2 P. M. A visit to Harvard Buildings, or excursion to the New England Gas & Coke Co., at Everett.
7.30 P. M. Banquet for members and ladies at Hotel Lenox.

THURSDAY.

9.30 A. M. Session of the Society at 6 Lowell Building.
1 P. M. Lunch at the Technology Club.
2 P. M. Afternoon session of the Society at 6 Lowell Building, or inspection of Technology buildings.

The following programme has been arranged for the ladies:

TUESDAY.

1 P. M. Lunch at the Technology Club.
2 P. M. Excursion to Wellesley College.
8 P. M. Presidential address.

WEDNESDAY.

1 P. M. Lunch at Harvard Union.
2 P. M. Inspection of Harvard Museums, including glass flowers.
7.30 P. M. Banquet at Hotel Lenox.

THURSDAY.

1 P. M. Lunch at the Technology Club.

The list of papers, as far as announced, is as follows:

A. B. Albro, "The Microstructure of Silicon and Silicon Alloys."
W. D. Bancroft, "The Rotating Diaphragm."
Lucien Ira Blake, "Electrostatic Method for Separating and Concentrating Ores."
C. F. Burgess, "Notes on the Economic Temperature of Copper Refining Solutions."
H. R. Carveth, "Electrolytic Potential of Chromium."
Ernest Fahrig, "Treating Low Grade Ore and Tailings by Electrolysis."
W. S. Franklin, "Reversible and Irreversible Polarization."
H. M. Goodwin, "On Billitzer's Method of Determining Absolute Potentials."
C. Hambuechen, "An Optical Method of Observing Diffusion in an Electrolyte."
William S. Horry, "An Electric Switch."
G. A. Hulett, "A Low Voltage Standard Cell."
L. Kahlenberg, "Specific Inductive Capacities."
A. Lodyguine, "Some Experiments on the Reduction of Titaniferous Ores Into Steel and Titanate of Iron."
A. Lodyguine, "Experiments on the Reduction of Different Oxides of Lead by Electric Current."
A. Lodyguine, "Experiments on the Electrodeposition of Metals on Aluminium."
H. E. Patten, "On the Heat of Solution of Aluminium Bromide in Ethyl Bromide."
A. Reuterdahl, "The Interdependence of Atomic Weights and Electrochemical Equivalents."
J. W. Richards, "Conduction in Electrolytes."
A. J. Rossi, "On the Utilization of Blast Furnace Gases in Connection with Electric Smelting."
R. C. Snowdon, "Electrodeposition of Silver."
R. C. Snowdon, "Adherence of Nickel to Nickel."
C. P. Townsend, "A Diaphragm Cell for Electrolysis of Brine."
E. Weintraub, "Mercury Arc."
W. R. Whitney, "Electric Arcs."
W. R. Whitney, "Colloids."

Notes.

American Electrochemical Society.—At the March meeting of the board of directors of the American Electrochemical Society, the following gentlemen were elected members: T. Herbert Arnold, State College, Pa.; L. O. Kuhn, State College, Pa.; James C. Armor, Pittsburg, Pa.; David B. Carse, New York City; W. S. Franklin, Bethlehem, Pa.; Max LeBlanc, Karlsruhe, Germany; Bunjiro Masiyima, Kokura, Buzen, Japan. At the April meeting of the board of directors, the names of the following gentlemen will come up for election: Daniel McMaster, Rumford Falls, Me.; Arthur D. Little, Boston, Mass.; Marten Hedlund, Haustahammar, Sweden.

Lehigh University.—The Register of Lehigh University, South Bethlehem, Pa., just issued, copies of which may be had on application, shows the attendance of 630 students from twenty-four States and eight foreign countries, the largest in the history of the institution. Seven students are taking the course in electrometallurgy. There are 56 members in the teaching staff. Thirteen four-year courses of instruction are offered at the university: The classical course, the Latin-scientific course, the courses in civil, mechanical, marine, metallurgical, mining, electrical, and chemical engineering, analytical chemistry, geology, physics, and electrometallurgy. A list of graduates of the university, with their present occupations, 1399 in number during the thirty-nine years of its existence, indicates that this institution is exerting a marked influence on the industrial development of the United States and of foreign countries.

Rensselaer Polytechnic Institute.—This institute was subjected to some inconvenience during the past year on account of the burning of the Main building in the summer of 1904. All the laboratories are in complete working order, however, and have been since last October. By the end of this year a first-class modern building for recitation and drafting rooms will be ready. Mr. Andrew Carnegie has recently given $125,000 for this building. The trustees have just purchased a beautiful piece of property adjacent to that now owned by the institute, for which they have paid $125,000. The Carnegie Building will be placed upon this. Although the existing chemical laboratory is finely equipped, it is becoming too small for the large number of students. This number is the largest in the history of the school. For this reason Mr. J. J. Albright, of the class of '68, has given $50,000 towards the erection of a new chemical laboratory, and one to cost $100,000 or more will be built during the next year.

Franklin Institute.—On March 15 Mr. Byron E. Eldred lectured before the Franklin Institute on his process of regulating high temperatures. This process was described in our December issue, 1904 (Vol. II, page 495).

Misprint.—Line 14 of the second column of page 90 of our last issue should read, "low in silver and lead," instead of low in silicon and lead.

Electrometallurgy of Iron and Steel at the New York Elec-

trical Society.—At the meeting of the New York Electrical Society, held March 15, the general subject of discussion was electrometallurgy of iron and steel. President Sprague referred in his opening remarks to some experiments made by him ten years ago, with an induction furnace for steel-making in New York City; these tests did not lead to any commercial result at that time. Dr. E. F. Roeber then read a paper on the manufacture of ferro-alloys in the electric furnace, and spoke briefly on the electrolytic production of steel facings for cuts and electrotypes, and on electrolytic refining of iron. A large number of ferro-alloys and metals of considerable value were exhibited. Among them were some very large and beautiful specimens of metals and alloys, free from carbon, made by the aluminothermic method of the Goldschmidt Thermit Co.; especially pure chromium metal, molybdenum metal 98 to 99 per cent Mo, manganese metal 99 per cent Mn, ferro-titanium 20 to 25 per cent Ti, ferro-vanadium 25 per cent V. The Roessler & Hasslacher Co. exhibited specimens of ferro-chromium and ferro-tungsten. Mr. F. M. Becket, of the Niagara Research Laboratories, exhibited samples of the following alloys and metals, mostly for high-speed tool steels, but two of the same being used for other special purposes: Ferro-chromium 70.50 per cent Cr, 0.75 per cent C; 62.61 per cent Cr, 0.20 per cent C; chromium-iron-silicide 54.36 per cent Cr, 21.10 per cent Si; ferro-vanadium 38.00 per cent V; iron-vanadium-silicide 20 per cent V, 20 per cent Si; ferro-tungsten 70.80 per cent W; molybdenum metal 96.51 per cent Mo; 0.22 per cent C; 92.71 per cent Mo, 4.40 per cent C; titanium carbide; and copper nickel 51 per cent Cu, 49 per cent Ni. Dr. Paul L. T. Héroult then presented a very interesting review of his work in connection with making and refining steel, which has already been fully covered in these columns. (See also the abstract of a paper of Combes in the Synopsis of this issue.) The discussion which followed was participated in by Messrs. J. T. Morehead, E. Stütz, L. Ruhl, Leonard Waldo, C. A. Doremus and C. S. Hurd.

Dr. Doremus spoke especially on the reduction of pig iron from ores, and pointed out that it is simply a question of the relative cost of electric power and fuel. He showed, however, that there are other considerations to be taken into account, for instance, that it is not necessary to use a certain quality of coke in the electric furnace. Dr. Leonard Waldo gave some reminiscences of early electrometallurgical work done in this country, and spoke especially on Mr. Colby's and his own work with the induction furnace, which he said is quite alive not only in Sweden, but in this country. Dr. Héroult, replying to several questions, thought that the induction furnace has a certain disadvantage in the necessity of using very pure starting materials and in the difficulty of cleaning. It was found that while Kjellin gets very good steels, they are not always exactly the same grade. Mr. C. S. Hurd referred to the possibility of getting cheap electrical energy for electric steel furnaces by utilizing the waste gases of blast furnaces in gas engines, coupled to electric generators.

The Occurrence of Tantalum Minerals.

By David T. Day, Ph. D.

The article on the tantalum lamp in a recent number of this journal has called such considerable attention to the question of a proper supply of this substance that a general review of the occurrence of tantalum minerals in the United States may be valuable at this time.

The following table gives a list of the localities where tantalum minerals have been recorded in various scientific publications. Recourse has been made, particularly to Dana's Mineralogy, the American Journal of Science, and the Transactions of the American Institute of Mining Engineers. The table also shows the percentage of tantalum which can be expected where the minerals are carefully selected.

Quite a different question is the amount of these minerals which can be available in any given locality and the percentage of tantalum which can be expected when the minerals are mined on a large scale and without any very great possibility of carefully cleaning each specimen. Taken altogether, it is evident that the following localities are most promising: Llano County, Texas, Mitchell County and other nearby localities where mica is mined in North Carolina, and the classic locality in Massachusetts from which columbite was first obtained.

In addition to these localities where tantalum minerals have long been known to occur, another source of this element seems fully as promising as any of the others. For many years the Pennsylvania Salt Co. has been importing cryolite into this country from Greenland. This cryolite is never pure, but contains galena and other lead minerals, spathic iron ore, fergusonite, sulphurettes of iron and copper, pachnolite, thomsenolite, arksudite, geoarksudite and hagemanite, and possibly columbite, tantalite, yttrolite, tapiolite and samarskite.

It is very probable that the residues from working up cryolite are still in existence in large quantities at the works of the Pennsylvania Salt Co., which will be valuble for extracting the tantalum. This may very possibly constitute an important source of the metal, where the quantity at hand can be easily measured and the percentage of tantalum readily determined.

Even very low grade material already mined, as is this, should afford an attractive source of the desired element.

In regard to the well-known localities where minerals containing tantalum have been found, it should be noted that fergusonite has been found with quite a number of other rare minerals at a point 5 miles south from Bluffton, Llano County, Texas, and considerable mining has been done at this locality for thorium minerals, so that it will be easy to determine just how much tantalum minerals can be obtained. The deposit here has the form of a mound made up of huge blocks and masses of quartz and red feldspar, as described by Messrs. Hidden and Mackintosh in the American Journal of Science for 1899. This mound has been entered with trenches on all sides, and it is quite easy to trace the occurrence of these rare minerals by the green staining from uranium compounds.

While many localities have been noted for the occurrence of columbiun and tantalum minerals in North Carolina, the neighborhood of Burnsville in Yancey County, and in the adjoining Mitchell County, is the most promising locality for further search. All mica mines in North Carolina should be explored very carefully for tantalum minerals.

Dr. J. Lawrence Smith has also found tantalum near Rockford, Coosa County, Ala., containing 79.65 per cent. of tantalic acid.

Rockport, Mass., is the classic locality from which the original specimens of columbite came which led to the discovery of columbium. While mineral interest in this locality has practically ceased, it should now be prospected again.

The Amazon stone of El Paso County, Col., also contains columbite, and a careful search of that locality is advisable. No doubt the energy with which uranium and vanadium and other rare elements have been sought and found in Colorado within the last few years will be applied also to the search for tantalum minerals with a fair prospect of success.

In the meantime, however, the quickest results may well be sought in the Black Hills of South Dakota, where, during last year, material said to be fairly pure columbite, to the extent of about 2500 pounds, was mined and sold to mineral dealers. Tantalite and other tantalum minerals occur with the tin ores in the Black Hills. It was recorded in the thirteenth volume of the Transactions of the American Institute of Mining Engineers, that Prof. Charles A. Shaeffer, of Cornell University, in 1884, examined some specimens of supposed tin stone, in which he was surprised to find a very slight trace

of tin. Further investigation showed the mineral to be tantalite and a considerable amount of the material was found. This came from the classic Etta tin mine, near Rapid City, South Dakota, which has since yielded such large specimens of spodumene in commercial quantity. Professor Shaeffer was unable to find any trace of columbium or tungsten in this material, and drew the conclusion (doubtless justified) that the Dakota tantalite is much simpler in composition than the samples from North Carolina and elsewhere.

In commenting upon this paper by Professor Shaeffer, Professor Blake, who had already noted the black tantalite in the tin ore of South Dakota, calls particular attention to two localities where the mineral tantalite is found in the same dike or vein as cassiterite; first, in the Etta mine, and, second, in the Bob Ingersoll claim. In the Etta mine it should be noted that the tantalite is found in close association with feldspar in crystals from one to four or five inches broad, and from one-quarter of an inch to three inches thick. It is not mixed with the cassiterite and it can be easily taken out. It is easily distinguished from the tin stone by the black color and the dark streak which is very different from that of tin stone. Professor Blake states that it is rarely found in the greisen rock (coarse granite) with the tin ore, although a few crystals have been seen near the outer margin of one of the irregular masses of tin ore. Professor Blake found one very remarkable mass of tantalite which he estimated to weigh over 2,000 pounds, though this was the only body of the mineral seen in that part of the vein. The mineral was found also in a "greisen-like aggregation of mica," not tin-bearing. Professor Blake did not find it in the tin ore veins of the Hill City district.

Prof. Frank B. Carpenter, now of Denver, Colorado, also reports in the Transactions of the Institute of Mining Engineers for 1888, Volume 17, the occurrence of large masses of tantalite or columbite in the Bob Ingersoll claim. His examinations, however, of specific gravity tests, make him pronounce this material columbite instead of tantalite, but he notes that tantalum is always present, but not to such an extent as to be a predominating element.

An analysis of the material from the Etta mine gave 79 per cent of tantalic oxide to Professor Shaeffer, but Doctor Headden of the Dakota School of Mines found only 18 per cent of tantalic oxide but 54 per cent of columbic oxide.

From the descriptions given above, it would seem entirely probable that an adequate supply of tantalum can be easily found for the Siemens-Halske lamp. Further, it is possible that the examination of the heavy sands of the Pacific Coast, to be undertaken by the Division of Mineral Resources of the United States Geological Survey, may yield further supplies of such materials.

Mineral.	*Sample from.*	*Percentage of Ta.*
Hatchettolite.		
Tantalo-niobate of uranium, near pyrochlore.	Occurs in Mitchell Co., N. C.	
Microlite.		
Essentially a calcium pyrotantalate. $Ca_2Ta_2O_7$.	Virginia	68
	Greenland, Ferg.	6
Fergusonite.	Ytterby, Sweden, wy.	27
Essentially a metaniobate (and tantalate) of yttrium with erbium, cerium, uranium, etc.	Ytterby, Sweden, brn.	8
	Ytterby, Sweden, brn.	10
	Helle, Bragite	2
	Karafvet, Yttr. gry.	43
	Burke Co., N. C.	4
Columbite.	Mineral Hill, Pa.	83/100
Niobate and tantalate of iron and manganese. $(Fe,Mn)(Nb,Ta)_2O_6$.	Standish, Me.	9
	Craveggia, Italy	13
	Isergebirge	16
	Branchville, Conn.	19
	Branchville, Conn.	52
Columbite.	Bodenmais, Bavaria	23
	Bodenmais, Bavaria	31
	Haddam, Conn.	29
	Amelia Co., Va.	53
	Northfield, Mass.	57
	Sanarka, Manganotantalite.	80
	Etta Mine, Black Hills, S. D.	18 to 53
	Elsewhere in Black Hills, S. D.	18 to 45
	Mitchell Co., N. C.	9
	Elk Creek, S. D.	34
	Nigger Hill Dis., S. D.	58
Tapiolite.		
Tantalate and niobate of iron.	Occurs in Sukula in Finland	74
Yttrotantalate.		
The so-called yellow yttrotantalate of Ytterby and Kararfvet belong to fergusonite.	Occurs at Ytterby in Sweden	46
Samarskite.		
Mineral related to samarskite has been found in granitic debris on Devil's Head, Douglas Co., Col.	North Carolina	18
	North Carolina	14
	Devil's Head, Colo.	27
	Devil's Head, Colo.	28
	Devil's Head, Colo.	19
Hielmite.		
Stanno-tantalate (and niobate) of yttrium, iron, manganese, calcium. Formula doubtful.	Occurs in Kararfvet.	54
	In Sweden	72
Polymignite.		
Niobate and titanate (zirconate) of the cerium metals, iron, calcium.	Occurs at Fredriksvarn and Svenor in Norway. Reported from Moravia and at Beverly, Mass.	1
Polycrase.		
Niobate and titanate of yttrium, erbium, cerium, uranium, like euxenite.	Occurs at Hittero in Norway, and at Slattakra in Sweden	4

CORRESPONDENCE.

Electric Induction Furnace.

To the Editor of Electrochemical and Metallurgical Industry:

Sir.—In view of the great interest which the splendid work of Mr. Kjellin, in Sweden, with the electric induction furnace for steel-making has aroused, I would like to call attention to three fundamental patents for the induction furnace which were granted to Mr. E. A. Colby (of the Baker Platinum Works, in Newark) in 1890. They are Nos. 428,378, 428,379 and 428,552, May 20, 1890.

The first claim of No. 428,378 reads as follows: "The combination with a closed primary circuit and means for supplying electric currents thereto and controlling the same of a refractory mass adapted to contain metal to constitute a secondary circuit and adapted to place such metal in inductive relation to the primary circuit, whereby electric currents may be induced therein by the action of electric currents in the primary circuit."

Patent 428,552 refers especially to the melting, refining and casting of metals in such a manner they will be free from blisters, pin-holes and occluded gases. A. N. H.

New York City.

Experiments with an Electro-Thermic Muffle Furnace.

By F. A. J. FitzGerald.

When a boiler has been designed and built it is a comparatively easy matter to make quantitative tests of its performance. From the amount of coal used in heating the boiler we can calculate the total energy used, and by measuring the amount of water evaporated and the steam pressure, the energy obtained in the form of steam can be determined. In dealing with electric furnaces the problem is different, for, while it is a very easy task to measure the energy put into the furnace, it is by no means easy to estimate how much has been used in useful work, and how much has been lost. In commercial work furnaces may be usefully compared by determining the amounts of power used and the output of the product manufactured. In this way the relative efficiencies of the furnaces may be compared, and so far as purely commercial work is concerned, the amount of manufactured material produced per kilowatt-hour is the important point. In making experiments with the electric furnace it would be of great advantage to have some method of determining the efficiency, so that calculations might be made as to what could be done with larger furnaces.

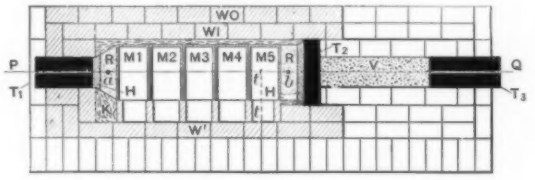


FIG. 1.—SECTIONAL PLAN OF FURNACE.

In what follows, there is a description of a muffle furnace built by Mr. Bennie and the writer in order to determine the behavior of such a furnace, and what could be expected of it in actual work. In addition to this a detailed account of certain experiments made with the furnace will be given. It will be very evident that both the construction of the furnace and the method of carrying out the experiments are crude in many respects. Nevertheless, the experiments yielded a certain amount of information that may be used as a guide in the construction of more elaborate furnaces and the carrying out of more refined experiments.

The total power available for the experimental furnace was 20 kilowatts, and this could be used either at 110 volts with 180 amperes, or at 55 volts with 360 amperes. It should be mentioned here that the power factor of the circuit is unknown, so that it may introduce a certain error into all the experiments, but as the latter were not very refined in their details the errors are probably of no serious consequence.

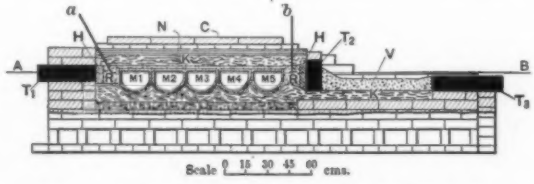


FIG 2.—PARTIAL SECTIONAL ELEVATION OF FURNACE.

In Fig. 1 is shown a sectional plan, and in Fig. 2 a partial sectional elevation of the furnace, the drawings being made to scale. The plan is a section through the line *AB* in Fig. 2, and the elevation a partial section through the line *PQ* in Fig. 1. The base of the furnace has a cellular structure and is about 12 cms. thick above the cells. The furnace contains the five muffles, *M1*, *M2*, etc., and is connected in series, with a regulating resistor. *T1* and *T3* are the main terminals of the apparatus, *T2* the terminal common to the furnace resistor *R*, and the regulating resistor *V*. The connections between the furnace resistor *R* and the terminals *T1* and *T2* are made at *H* with graphite. The resistor has two branches, one passing over the top, the other at the bottom of the muffles. The bottom of the furnace is filled with magnesite *O* and the muffles with the resistor embedded in amorphous silicon carbide *K*. At the top of the furnace the silicon carbide is covered with four layers of stout asbestos board, and finally four large tiles *C* are laid on top, resting on the walls of the furnace; *a* and *b* are carbon rods to which voltmeter connections can be made. All the muffles except *M5* were filled with sand. A porcelain tube passed into *M5* at *tt'*, so that the inside could be observed during the working of the furnace. The regulating resistor consists simply of a trough filled with granular carbon *V*.

In the following table the dimensions of various parts of the furnace are given:

Furnace:		
Distance between terminals (*T1* and *T2*)	145	cms.
Total width	91	"
Thickness of wall *WI* + *WO*	23	"
Thickness of *K* between muffles and *WI*	4	"
Distance from open end of muffles to *W'*	18	"
Thickness of *K* above muffles	7	"
Thickness of *K* below muffles at thinest place	5	"
Thickness of furnace bottom	7	"
Thickness of *O*	5	"
Muffles:		
Length	38	"
Width	23	"
Height	16	"
Resistor:		
Width	38	"
Length of upper branch	115	"
Length of lower branch	140	"
Heating surface above muffles	43.7	dcm.²
Heating surface below muffles	53.2	"
Thickness of lower resistor	4	cms.
Thickness of upper resistor	varied	
Regulating resistor:		
Length	88	cms.
Width	21	"
Thickness	varied	

Before building the furnace described above a similar furnace was built with only three muffles, in order to form an estimate of the energy that would be required to obtain in the muffles a temperature of at least the melting point of steel. The preliminary experiments with this small furnace indicated that 100 watts per square decimeter would be ample. The total heating surface of the resistor amounts to 97 square decimeters; therefore, the furnace should require about 10 kilowatts.

From the preliminary experiments it was also determined that a total thicknss of 8 cms. for the resistor would give the proper resistance. Here, however, a difficulty arose, for, on account of the shape of the muffles the lower resistor had a greater length than the upper resistor, hence, in order that the quantity of energy developed in each should be the same, the thickness of the resistor would have to be inversely as their lengths. Thus, if the lower resistor is 4 cm. thick, the upper resistor should have a thickness of

$$\frac{115 \times 4}{140} = 3.2 \text{ cms.}$$

But another consideration arises at this point, for it was desired to heat the muffles as uniformly as possible, therefore, the number of watts per square decimeter should be the same, both above and below the muffles. It follows then that since the energy developed in the resistors will be inversely pro-

portional to their resistances, the latter should be inversely proportional to the heating surfaces. If r_1 is the resistance of the upper, and r_2 the resistance of the lower resistor, then

$$\frac{r_1}{r_2} = \frac{53.2}{43.7} = 1.22$$

Therefore the proper thickness for the upper resistor would be

$$\frac{3.2}{1.22} = 2.6 \text{ cms.}$$

Still another complication exists, for hitherto we have tacitly assumed that the resistivities of both resistors would be the same. This, however, will not be the case, for the resistivity of the lower resistor will be less than that of the upper resistor, because of the greater pressure on the former by the muffles and their contents. This effect cannot be calculated, so it seemed best to leave the determination of the size for the upper resistor to experiment. Alterations in the upper resistor could easily be made by removing the top of the furnace.

In Fig. 3 there is shown a general view of the 3-muffle furnace. The three bricks projecting from the side wall of the furnace indicate the position of the muffles, as these could be removed so that the interior of the muffles might be inspected. The bottom of this furnace was only 7 cms. thick, and it was observed that after a run of a few hours the loss of heat through the bottom was very great, hence, a considerably greater thickness was used in building the next furnace. The first stage in the construction of the furnace now under consideration is shown in Fig. 4. The lower resistor is in place, and the five muffles, the openings into the latter being shown. The scoop is lying on the top of the muffles. In Fig. 5 is shown the same side of the furnace as in Fig. 4 after the building was finished. In Fig. 6 the other side of the furnace is shown with the rod *b* projecting and connected with the voltmeter. At the left side is shown the regulating resistor with bricks laid on the granular carbon to decrease the resistance.

FIG. 3.—GENERAL VIEW OF FURNACE.

After the furnace was built some preliminary experiments were made to find out the best method of manipulating the regulating resistor *V*, so as to obtain the desired generation of energy in the furnace. In order to observe the temperature in *M*5, a thermometer reading to 300° C. was introduced by means of the porcelain tube at *t*. When the temperature in the muffle was too high for the thermometer a carbon rod with a little cup-shaped hollow bored out at one end was put into the porcelain tube and in the hollow a small piece of metal was placed, so that the time of its fusion could be observed. The

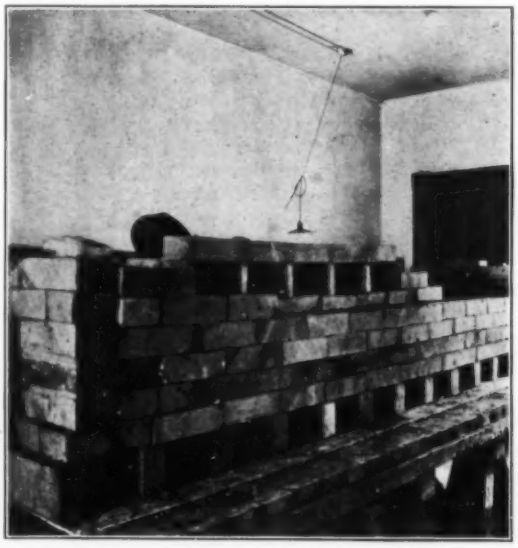


FIG. 4.—FIRST STAGE OF CONSTRUCTION.

preliminary experiments were made one afternoon and only lasted for about one hour. The temperature indicated by the thermometer in *M*5 at the end of the experiment was 113°. Next day an experiment was begun in the morning and lasted for nearly 9 hours. Voltmeter and ammeter readings were taken every 15 minutes or oftener, and from these the kilowatts used and the resistance of the resistor calculated. The results are shown in the curves of Fig. 7, where the full-line curve shows kilowatts and the broken-line curve resistance.

FIG. 5.—FINISHED STRUCTURE.

The temperatures are given at various points on the kilowatt curve, the figures representing degrees centigrade and the word "lead" indicating that lead melted at this point.

Considering first the run of the furnace up to 7.75 hours, it

is found that the mean ordinate of the power curve is 3.84 kilowatts, which gives approximately 40 watts per square decimeter of heating surface. At 7.75 hours the inside of $M5$ just showed a dull red heat, but the temperature was not high enough to fuse aluminium, introduced by means of the carbon rod. No doubt, in time, aluminium would have fused and the temperature of the muffles and their contents would have reached at least a good red heat, but, except for special purposes, this gradual heating would not be desired. The experiment showed that the regulation of temperature was well under control.

At 7.75 hours the voltage at the main terminals was increased to 110 volts, and the resistance of V increased. The result of this is shown in the curves, and it is somewhat surprising, for the resistance of the furnace resistor is increased by 21 per cent at once, and for three minutes more continued to increase until it was 47 per cent greater than before. After that it decreased, and would undoubtedly have reached its original amount if the experiment had been continued. This

FIG. 6.—OTHER SIDE OF FURNACE.

curious phenomenon is explained by a characteristic of granular carbon resistors.

If a current is passed through a loosely packed granular carbon resistor hundreds of minute arcs are usually formed at first between the carbon grains. After a time, as the resistor becomes hot, these disappear and the crackling noise accompanying the arcs ceases. Apparently, after heating, somewhat better contact is made between the grains, and there is no longer any arcing. Now suppose that there are 50 volts across the terminals of the resistor, and when the arcing has ceased the potential is suddenly increased to 100 volts, it will frequently happen that the crackling noise, due to the formation of the arcs, will begin again and after a time die down when better contact is made. Accompanying this phenomenon is an increase in the resistance of the resistor.

This is what happened in the experiment described above. At the beginning of the run there were 55 volts between terminals $T1$ and $T3$; then the resistance of V was gradually decreased by piling bricks on the granular carbon till 6.25 hours, when the e. m. f. between a and b was 42 volts. No further change was made till 7.75 hours, when, after increasing the resistance of V, the e. m. f. between $T1$ and $T3$ was increased to 110 volts. This increased the e. m. f. between a and b to 65 volts, causing the temporary increase in resistance observed.

In building this furnace the upper branch of the resistor was 4 cms. thick; that is to say, it was made of the same thickness as the lower resistor, in spite of its being shorter and having a smaller heating surface. The observations made in $M5$ showed that although the upper resistor was so thick the generation of heat in the lower resistor was more rapid, for when the muffle first showed a red heat this was noticed in the bottom. This showed that the effect of the pressure of the

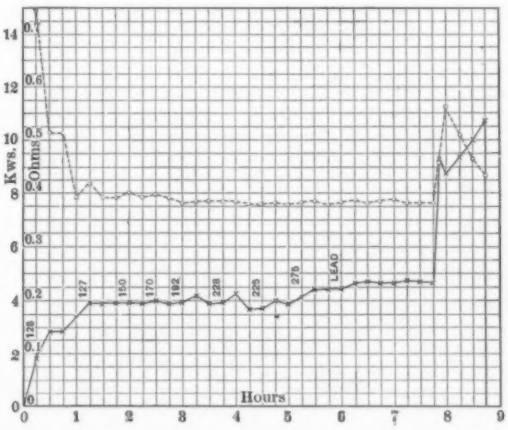


FIG. 7.—CURVES OF POWER AND RESISTANCE.

muffles on the lower branch of the resistor made the resistance lower than that of the upper branch.

Although this uneven heating was observed, it was decided before making any change to try another experiment, using more energy. The experiment was made on the day following the previous run, and the furnace started with a temperature well over 100° in $M5$. The results obtained are shown by the curves in Fig. 8. When silver melted in $M5$ at 3.25 hours, the power was cut down with the object of finding out whether, having reached this temperature, the furnace could be held at that point with a much smaller consumption of energy. It was found, however, that the furnace cooled off rapidly, and although at the end of an hour $M5$ still showed a red heat,

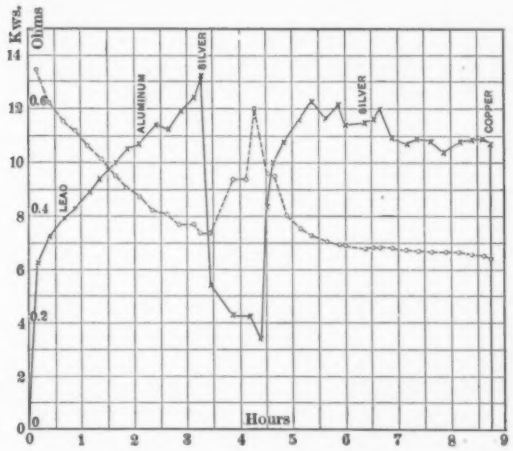


FIG. 8.—CURVES OF POWER AND RESISTANCE.

silver would no longer melt, and after increasing the power it was two hours before a melt was again obtained.

The resistance curve of this furnace is instructive. When the power was cut down there was at first no change, but presently the resistance began to increase rapidly, due to the cooling of the furnace. It will be observed that the resistance increases to close on the value that it had when the experiment was started, although there can be no doubt that the tempera-

ture was much higher than at first. This is due to the formation of carbides in the resistor, the carbides acting as fairly good conductors, while very hot. There was therefore a permanent increase in the resistance of the resistor. The effect was not produced after the first experiment, because the temperature of the resistor was never high enough to form carbides. This explanation of the change in resistance was confirmed later, when the upper branch of the resistor was exposed, for the temperature had been high enough to form a quantity of amorphous silicon carbide on the carbon grains. It is always important to keep in mind the probability of forming carbides when experimenting with resistance furnaces of this kind, for their formation frequently explains effects which at first appear to be somewhat puzzling.

The fact that carbides had formed in the upper branch of the resistor showed that its temperature was far above that observed in $M5$, for amorphous silicon carbide does not form till a temperature well above that of molten steel is reached. The temperature conditions in the other muffles were observed during the run by thrusting copper wires through the interstices of the wall into the sand in the muffles. These observations showed that the temperature in these muffles was higher than that of $M5$ for copper fused in them before fusion was obtained in $M5$. The spaces between the openings of these muffles and the wall W' being filled with sand there was less loss of heat than from $M5$, where the muffle and the space in front was empty.

Before making the next experiment the upper branch of the resistor was uncovered and some more carbon put on it, so as to make it thicker, thus reducing the resistance. The results obtained in the experiment are shown in Fig. 9.

Although the upper branch of the resistor had been made thicker, the resistance was now much higher than before, because of the carbide formation. It had been hoped that with this increase in the thickness of the upper branch of the resistor, the resistance would be lowered sufficiently to make a

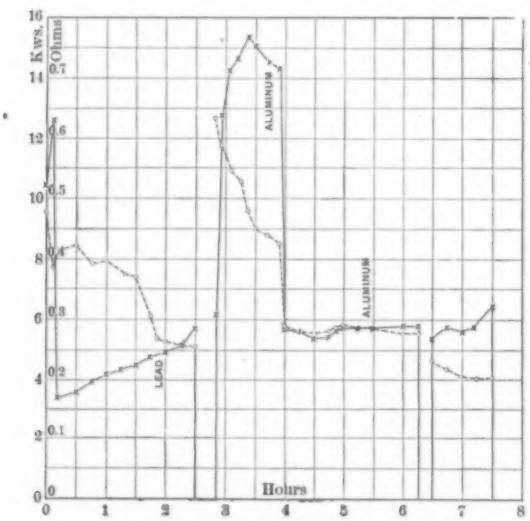


FIG. 9.—CURVES OF POWER AND RESISTANCE.

run with an e. m. f. of 55 volts between $T1$ and $T3$, but evidently this was not possible, as enough energy could not be developed in the furnace using this low voltage. Accordingly, it was decided to try another experiment. The current was cut off, and after removing the tiles from the top of the furnace and taking off the sheets of asbestos, ten bricks were laid on the amorphous silicon carbide K. When the current was thrown on it was found that instead of a decrease in resistance there was an increase. Evidently in laying the bricks on K some of the amorphous silicon carbide had been forced down into the resistor, thus increasing its resistance. However, with 110 volts at the terminals, the furnace was rapidly heated up and the melting point of aluminium reached in $M5$.

It will be remembered in the experiment of Fig. 8 it was found that when the power was cut down the furnace cooled off rapidly, and it was some time before the temperature could again be raised to the point it was at when the cut was made. In the present experiment another attempt was made to cut down the power and try to hold the temperature constant at the melting point of aluminium. The e. m. f. between the terminals was therefore changed from 110 to 55 volts, and the watts in the furnace held at about 5.5 kw. As nearly as could be observed, this held the temperature, which would indicate that this rate of developing heat was about sufficient to supply the losses by radiation. This seemed to be a serious loss, and indicated that the heat insulation of the furnace was very defective.

The experiment also showed that the upper resistor was unsatisfactory, since its resistivity could be so easily varied by the granular amorphous silicon carbide which covered it. This difficulty might probably have been overcome by covering the resistor with some material in the form of tiles; but nothing of this kind being available at the time, it was decided to use a resistor composed of a mixture of coarse and fine granular carbon, so that it would form a mass, into which the amorphous silicon carbide would not sift. It has been shown elsewhere that as a general rule in making resistors, it is better to use a carefully graded carbon, as otherwise it is difficult to get a homogeneous structure. The present case, however, is a special one, for the resistor is in the form of a wide, thin plate, and it was therefore easy to obtain a homogeneous structure with a mixture of coarse grains and powder. A granular carbon resistor made in this way has much lower resistivity than a resistor formed of coarse grains alone, and experiment showed that the proper thickness was about 1.5 cms.

An examination of the granular carbon composing the upper resistor in the previous experiment showed that it contained considerable quantities of amorphous silicon carbide, which formed a coating over the surface of the carbon granules. The surface of the muffles also showed that the refractory material of which they were composed had been superficially fused, and it was undoubtedly from the silicon contained in them that the amorphous silicon carbide coating the grains of the resistor was derived. As Moissan has shown, silica has an appreciable vapor tension below its temperature of fusion, and when actually fused vaporization is comparatively rapid. This having taken place in the upper resistor it was plain that the same thing must have happened in the lower resistor and consequently it was not surprising to find an increase in its resistivity.

The results obtained from the final experiment are shown in Fig. 10. At the end of 4.75 hours copper was at once fused when introduced into muffle No. 5, and when pieces of iron wire were thrust into the sand contained in the other muffles they were instantly fused. About this time it was observed that the readings of the ammeter became very unsteady, and the curve shows that the resistance of the resistor was varying considerably. This made it evident that some serious disturbance was taking place in the furnace, the most probable explanation being fusion of the muffles. Meanwhile, the temperature in $M5$ had diminished, and just before cutting off the current at the end of the experiment copper would no longer melt. The fusing of the muffles would greatly increase the resistance at the point where the fusion took place, the generation of heat at that place would be increased, diminishing the generation of heat at mufflle No. 5, and consequently allow it to cool. When the furnace was taken down after cooling, it was found that two of the muffles had fused, as shown in an illustration to a previous article (Electrochemical Industry, Vol. II, No. 12, page 439).

An examination of the granular carbon of the resistor showed that at the muffles where fusion had taken place the temperature had just reached the point of the formation of crystalline silicon carbide.

From this experiment, as well as from those that went before, it was plain that the heat insulation of the furnace was very far from being sufficient. At the end of the run the bricks covering the cells at the bottom of the furnace were red-hot, as were the pieces of asbestos board on the top of the furnace. The side walls were also very hot, being well above 100° C. A further evidence of defective heat insulation was found in the muffles, for it was observed that while the sand in the center parts of the muffles had been heated well above the melting point of iron, the temperature of the sand at the ends of the crucibles had only reached the melting point of copper. In the experiment of Fig. 9 it was shown that about 5.5 kilowatts were used to supply the heat lost by radiation from the furnace, since with that rate of generation of energy the temperature remained stationary, and in the experiment now considered, probably a much larger amount of energy was dissipated in this way, since the temperature was higher.

It was the evidence of this loss of heat that led to the use of a cellular bottom to furnaces having the cells filled with some

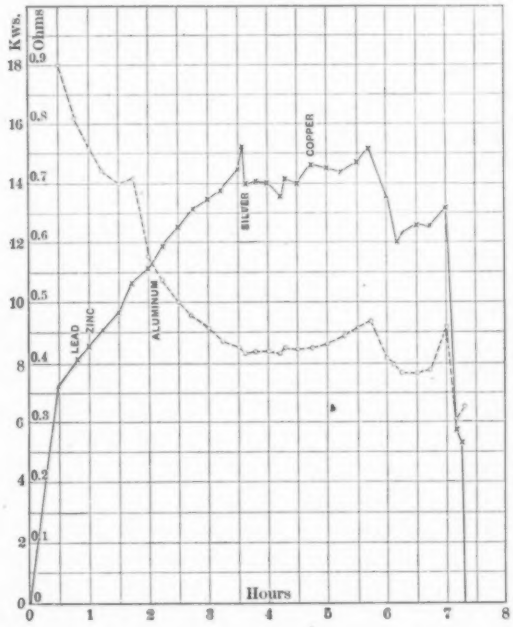


FIG. 10.—CURVES OF POWER AND RESISTANCE

good heat insulator, such as infusorial earth. The design of such a furnace bottom is shown elsewhere (ELECTROCHEMICAL AND METALLURGICAL INDUSTRY, Vol. III, No. 2, page 55), and has been found to effect a great saving of heat. With the same object, the side walls should, when possible, be double, with the space between the walls filled with a heat insulator. Finally, the top of the furnace should be insulated in a similar manner. A close attention to heat insulation is of less importance when dealing with much larger furnaces, since the ratio between the superficial area of the furnace and its volume is much less than in the small apparatus, but it is doubtful if sufficient attention is directed towards this subject.

The calculation of the efficiency of an electric furnace is always very difficult, as was pointed out before, because of the lack of trustworthy data on which to base the estimates, but an attempt will be made to obtain some notion of the efficiency of the furnace described above.

In this furnace the useful work will be considered as the heating of the muffles and their contents. The main constituent of these materials may be assumed to be silica in the form of quartz, and a conservative estimate will place their increase of temperature at 1100° C. in the last experiment at the end of six hours' run. We then have the following data:

Mean ordinate between 0 and 6 hours, 12.14 kw.

Kilowatt-hours, 72.84.

Silica raised 1100° in temperature, 250 pounds.

Mean specific heat of quartz, 0.3.

Total heat units given to silica, the heat unit being that required to raise 1 pound of water 1° C., was $1100 \times 250 \times 0.3 = 82{,}500$.

Total heat energy developed in the furnace was

$$1890 \times 72.8 = 138{,}000.$$

Hence, the efficiency of the furnace was approximately 60 per cent.

The figure used for the specific heat of quartz is based on Pionchon's determination for quartz between 400° and 1200°. Other observers working at lower temperatures give other figures varying from 0.2 to 0.3. If the figure taken is 0.25, instead of 0.3, the efficiency of the furnace would be 50 per cent. The important point to observe, however, is that the efficiency is decidedly low, and shows very clearly the faulty design of the furnace.

It is, of course, obvious that if the muffles were unloaded while the furnace was running, and then loaded again, the efficiency would be very much higher for the next heat, for it would not be necessary to expend a large amount of energy in raising the temperature of the furnace itself.

In order to heat a muffle furnace of this kind rapidly it is necessary to use a very high resistor temperature, and to do this some other material than that used in making ordinary muffles must be employed. For this purpose silico-carbide is indicated. One of the great advantages of electrical heating is lost if we cannot push the temperature of the resistor to a very high degree, thus saving time in heating the material. It is, of course, obvious that in building an electrical muffle furnace along the lines of that described, the muffles should have a square cross-section, as the building of a curving resistor is by no means easy.

In the last experiment, if we calculate the watts per square decimeter of heating surface, from the value of the mean ordinate, we find that they amount to 125. With a furnace of better design as regards heat insulation, it is evident that 100 watts per square decimeter would be ample where the object was to raise the temperature of the muffles and their contents to the melting point of steel.

The granular carbon resistor as used in the furnace showed some objectionable features. It was evidently very susceptible to mechanical influences which caused great changes in its resistance; but a still more serious objection was the increase in resistance produced by the formation of carbides. This trouble will always be experienced where there are silicious bodies at a high temperature in proximity to the resistor. Probably the difficulty could be overcome to a great extent by using a refractory substance such as magnesia in contact with the resistor, since magnesia will not form carbide by heating with carbon to a high temperature. Some experiments made following this idea seem to be promising, but have not yet advanced very far. Finally, the experiments with the furnace suggest that solid resistors would have certain marked advantages; but a discussion of these must be reserved for another article.

Hittorf's Migration of Ions.—In *Science*, of January 27, F. H. Getman describes a model for explaining in the lecture room Hittorf's conception of migrating ions. The cations and anions are represented by black and white balls, respectively, sliding on wooden rods. Above the rows of balls a scale is provided to measure the speeds.

Refractory Materials for Furnace Linings.

There is scarcely a detail in metallurgical furnace practice which requires greater attention than the provision of a refractory furnace lining. According to the temperature of the furnace, different furnace linings will, of course, be used. Materials specially useful for the temperatures in electric resistance furnaces were discussed very fully in an article of Mr. F. A. J. FitzGerald in our November issue, 1904, page 439.

The subject of refractory materials for furnace linings has recently been the subject of an able paper by Mr. E. Kilburn Scott before the Faraday Society. In the following we give an abstract of this paper.

The author divides refractory materials into:

I. Carbon for the highest temperatures.

II. Silicon-carbons made in the electric furnace, such as carborundum, or crystallized silicon-carbide, siloxicon and amorphous silicon carbide.

III. Magnesia made in the electric furnace.

IV. Ordinary fire brick, magnesium brick, etc., for the lower ranges of electric furnace temperature.

The author deals with the second and third of the above groups. He emphasizes that it is not necessary that the refractory linings should be homogeneous throughout, but that such ordinary materials as firebrick, magnesia brick, etc., may be washed over a depth of half millimeter or so, with a highly refractory material with greatly improved results.

Carborundum, Amorphous Silicon-Carbide, and Siloxicon.

In discussing first the use of carborundum as refractory material, Mr. Scott says that the corborundum is ground up very fine and mixed in the proportion of three parts by weight of carborundum to one part by weight of silicate of soda (water-glass). After thoroughly brushing the newly set fire brick to get rid of dust etc. (the mixture will not stick to a surface which has been already fired), the carborundum is painted on to the depth of about half a millimeter. It is then left for twenty-four hours to dry, and afterwards the firing started up gradually.

The result is that the fine particles of carborundum become cemented over the whole surface of the fire brick lining, cracks and all, and if properly done, it adheres quite soundly.

Carborundum taken from the core of the electric furnace is infusible at 7000° F., and is unaffected by oxygen, ozone, or sulphur at 3000° F. It is also unaffected by chlorine, bromine, or boiling concentrated sulphuric acid, gaseous hydrochloric, nitric, or hydrofluoric acid.

To show how this coating will resist high temperatures, Mr. Scott mentions that during the Düsseldorf exhibition, a welding furnace with coke firing and blast used for the welding of tubes worked daily from 9 o'clock until 7 at night for nearly six months, and the carborundum lining was practically in the same condition at the end as at the beginning of the run.

Where basic slags or basic materials have to be taken into consideration, fire clay is employed as a binder, instead of the water-glass, the proportion being usually six parts by weight of carborundum, to one part by weight of fire clay.

Carborundum, mixed with fire clay in equal parts, has been used for repairing cracked gas retorts in Berlin with considerable success. This was done whilst they were still hot, and they ran for a considerable time afterwards, perfectly gas-tight.

For pointing the joints in brickwork of coke ovens, for recovery of by-products, a mixture of carborundum and fire clay has been found most useful. The mixture is tamped in to about half an inch deep, and it completely closes the oven chamber against the flues, so that there is no entering of gases through the heating walls.

Carborundum has also been used for lining iron and other metals. In this case, three parts by weight of carborundum are mixed by weight with two parts of silicate of soda. Where it is used for this purpose, however, the metal must remain constantly heated, as otherwise the coating will burst off when the metal cools down.

Carborundum coatings have proved very suitable for coating iron, etc., which is subjected to considerable heat; thus, in the case of some central iron pipes for a particular blast furnace, their life was increased from two and one-half to nine months.

The temperature limit to the use of carborundum is reached when it decomposes according to the equation:

$$SiC = Si + C$$

the silicon escaping as vapor, while the carbon remains as graphite. When the temperature is so high that this decomposition occurs, there appears to be no refractory material that can be used, except carbon, and the best form of carbon for the purpose is charcoal, since its electrical resistivity is so high. Possibly further study of electric furnace products may produce other highly refractory materials.

Mr. Scott exhibited various samples, among them fire-clay bricks with a carborundum surface, to be used at the highest possible temperature; also, a piece of Glenboig fire brick coated with carborundum and taken from a furnace heated with coal dust firing; the brick was exposed to a very high temperature, but the carborundum surface had protected it in an exceedingly good way.

Mr. Scott also showed an injector furnace, one-half of which was coated with carborundum. This furnace was heated with a blow-pipe, whereby the uncovered part commenced to flux, while the corborundum covered part remained perfectly intact. A fire clay crucible coated with carborundum, which had been exposed to oxy-hydrogen blow pipes, was exhibited to show how efficiently carborundum had acted as a protector. A carborundum crucible made from 90 per cent carborundum and 10 per cent fire clay, samples of carborundum powder for refractory purposes, and carborundum crystals as obtained from the electric furnace, were also exhibited. Mr. Scott then gave a synopsis of the properties and applications of amorphous silicon carbide and siloxicon. Concerning the same, the reader may be referred to the article of Mr. FitzGerald quoted above.

Magnesia.

In a communication published on page 454 of our November issue, 1904, Mr. Scott referred to some experiments made by him on electrically-shrunk magnesite and some remarks were made on the same subject on page 455 by Mr. F. A. J. FitzGerald. Mr. Scott now deals on this subject at greater length, and for this reason this part of his paper is given in full as follows:

"Most of the magnesite at present used comes from Styrie in Austria; the Island of Euboea, in Greece; from California, and Salem in Southern India. The ordinary calcined mineral, generally in the form of brick, is now recognized as the best material for lining basic open-hearth furnaces, cement kilns, etc. It may be employed to advantage wherever high temperatures and chemical reactions are usually detrimental to dolomite, chromite, and silica. America uses practically all calcined magnesite for steel furnaces, whereas in England dolomite is chiefly used, while magnesite is coming in, although it is double the price. The distinctive characteristics of a calcined magnesite lining are durability, freedom from moisture and silica and resistance to corrosion when exposed to the action of basic slags and metallic oxides. These qualities make the lining cheaper than most others in the long run.

"The usual method of dealing with the crude magnesium carbonate which is found in a more or less pure state is to drive off the carbonic acid gas as far as possible by calcining, in a kiln. The kilns of the Anglo-Greek Magnesite Co., on the Island of Euboea, were originally roughly built kilns fired with wood, but now modern shaft calciners are employed fired with lignite coal. The temperature, however, is not sufficient to thoroughly shrink the magnesium carbonate, and it is even

doubtful whether rotary kilns would do it satisfactorily.

"About two years ago the writer carried out some original experiments in Meraker, Norway, with a view to thoroughly shrinking the magnesite in an electric furnace. The furnaces were of the Siemens & Halske type, built for the manufacture of carbide of calcium on the intermittent system, and they were supplied with current from 900-kw. three-phase generators, driven by water turbines. The material on which the writer worked was crude magnesite, brought from Salem, in Southern India,* which is found in a remarkably pure state.

"The writer found that it was possible to simply pass the magnesite down the chutes in the same way as the raw materials for making carbide of calcium. When once the arc was fairly started the raw magnesite could be fed in at a rapid rate and completely shrunk, that is to say, all the CO_2 driven off, without the assistance of any carbon or other material.

"After a two hours' run with 3500 amperes at 65 volts, a block about 15 inches cube was obtained which was practically pure crystallized magnesite. When broken into pieces with a sledge-hammer the material showed a fine crystalline structure with a beautiful iridescent color, and it had evidently been in a semi-plastic condition. Attempts were made to mould the half-shrunk material into bricks and then treat these in the electric furnace, but with poor success.

"On returning to England in November, 1902, the writer found that Mr. Deighton, of the Deighton Patent Flue & Tube Works, Hunslet, was trying carborundum as a wash in the furnaces, and also on the refractory blocks surrounding the water-gas flame when heating the steel tubes. I showed the electrically-shrunk magnesite (magnesia) and suggested that it would probably be as good and considerably cheaper then carborundum.

"This electrically-shrunk magnesite has been tested as a wash over the fire brick lining of a carbide of calcium furnace, and it was found that the bricks lasted for 200 hours without repair, whereas the unprotected bricks required repair after a five hours' heat. There is no doubt that the crystallized material, without further preparation than being crushed to suitable dimensions, will prove of special value as a refractory material in metallurgical practice, and an important point in connection with its use as linings for electric furnaces is that magnesia, unlike lime, does not form a carbide with carbon.

"In the large open-hearth steel furnaces at the Homestead Steel Works, Pittsburg, the writer found that large quantities of ground calcined or partly shrunk magnesite were used for the underbeds of the furnaces, and it is probable that electrically-shrunk material would be still more effective.

"In replying to a letter by the writer in ELECTROCHEMICAL INDUSTRY for November, 1904, Mr. FitzGerald states that he considers that the electrically-shrunk magnesite is not as refractory as silico-carbides, or carborundum, and he instances a case of a furnace in which calcined magnesite was used for part of the lining. The temperature at the inner surface of the magnesia lining was well below that of the formation of the carborundum, but above that of the formation of the silico-carbides. The heat lasted about eight hours, and the lining, which was about 4 cm. thick, was completely fused. Probably the reason for this was because Styrian magnesite, which is so largely exported to America, was used. This magnesite unfortunately contains a somewhat high percentage of iron, and this causes it to flux easily. Further, the magnesite was not crystallized, but only partly burnt.

"A pure magnesite, such as the Indian, when crystallized certainly does not run at the electrical furnace temperature. In the writer's experiments at Meraker it was never more than in a semi-plastic condition, even the temperature obtained in the thermit compounds had no effect on it."

Mr. Scott gives the following report of his test made on October 29, 1902, to produce electrically-shrunk magnesite:

"Twelve bags of calcined magnesite screened through a sieve with meshes about half an inch by a quarter, and the material then fed into the chutes leading down to the furnace. Started furnace with a shovelful of charcoal, and allowed some magnesite to run in. The material took longer to heat up than in making calcium carbide, but in about fifteen minutes was able to raise the top electrode, which showed that it was going to be possible to treat the material *without* coke. Later on, found it was much better to work without coke.

"In three-quarters of an hour the handle raising the electrode had been given 8½ turns, which meant that it had been lifted 5¼ inches, and in one hour it had been raised 7½ inches.

"The process was continued for two hours, by which time all the twelve bags had been used up and the furnace filled to within 6 inches of the top. The furnace was left to cool, and on the following day the hard core immediately between the electrodes was taken out and weighed about 3 cwts. and measured about 15 inches cube.

"The operation throughout was very quiet, this no doubt being partly owing to the absence of coke. Occasionally, when the material ran short and the edge of the electrode became uncovered some material would be thrown up with considerable violence.

"On breaking the block with a sledge-hammer the center and bottom was found to be a very dark gray, leading off to white on the outside of the block. There was also a considerable amount of yellowish white fused material and some reddish brown.

"Throughout the two hours the voltage was 65 volts, and the current kept as near 3500 amperes as possible.

"To make 3 cwts. of crystallized magnesite took about 200 horse-power for two hours."

* These magnesite deposits have been known to the natives for many generations, and they use it for plastering the inside walls of houses. Being absolutely fireproof and self binding, it is an excellent material for the purpose. It has been suggested to make partition walls for houses from it by smearing onto a textile fabric as a base, and letting it set in a corrugated shape. The samples the writer has seen appear to promise well.

Electrolytic Treatment of Electrolytic Slime.

BY ANSON G. BETTS.

Electrolytic slime has not a very pretty name, nor is it a very pretty material to look at or smell of, but it is the final mixture of everything known to the metallurgical world, and it presents a very pretty problem to the chemist, who has it forced on him to treat it for next to nothing, separate the worst possible tangle of metallic elements, save everything contained in its most valuable and marketable form, not lose any of it, do it in a hurry, without any complication in plant. But it has been found possible to meet all these requirements by electrochemistry. Electrochemistry seems destined to supplant almost all inorganic chemical and metallurgical processes, but only by the most painstaking effort. We need chemicals to make chemicals and metals, and they can be handled in no way as cheaply as electrochemically. Even as cheap a chemical as sulphuric acid, at five-tenths of a cent per pound, which will cover the cost of production and a fair profit, can be moved from one combination to another, allowing two volts, for about one-half kilowatt-hour, worth at a water-power, say, two-tenths of a cent.

Fusion processes have been used for slime, but the separations cannot be as complete as with wet processes, where we can deal with such individual properties of the different elements as electromotive forces of solution and solubilities in different solvents, and even practical requirements leave us some choice in the use of these properties. Fusion also causes loss by volatilization and absorption.

It was my intention to give a table showing all the wet processes tried, to illustrate their possibilities and objec-

tions, but it would occupy a good deal of space and not be interesting to most readers. For several years past I have been working on this problem, with the assistance of Dr. Edward F. Kern, and I think I have practically exhausted the possibilities of electrochemistry in this small field, and have gained in the laboratory the experience that could only be gained in a works in a much longer time, for it has been easy to make changes and try processes in a few hours that would take as many days or weeks in a plant. No scientific

The particular reactions of the finely divided metals as they exist in slime, with which we are here interested, are about as follows:

Lead, with sulphates and oxidizing agents is changed, of course, to insoluble lead sulphate. In the final fusion with soda, lead sulphate is easily fluxed.

Readily changed by HCl in a current of air, chlorine, ferric chloride, etc., to sparingly soluble lead chloride; not dissolved by sodium hypochlorite, sodium chloride solution, or

TABLE I.—SLIME ANALYSES.

No.	Anodes.	% Cu.	% Ag.	% Sb.	% As.	% Pb.	% Bi.	% S.	% Fe.	Oz. Au.	% Se.	% Te.	
1	Lead, Trail, B. C.....	8.83	28.15	27.10	12.42	17.05	nil	.....	1.27	.....	.....	....	*Trans. Am. Inst. Min. Eng.*, 1904, p. 182.
2	" "	22.36	23.05	21.16	5 40	10.62	nil	.....	1.12	.....	.....	....	
3	Lead, Monterey, Mex	1.90	32.11	29 51	9.14	9.05	trace	.....	.49	29.1	.....	....	*Trans. Am. Inst. Min. Eng.*, 1904, p. 183.
4	Lead, Mexican.....	9.30	4.7	25.32	44.58	10.30	.52	.....	.....	.....	.....	....	
5	Lead, Mexican.....	6.38	3.90	50.16	15.23	5.30	19.74	nil	nil	.....	.....	.....	Original.
6	Lead, Trail, B. C.....	1 40	31.62	35.71	4.91	9.57	.....	.....	.....	180.33	.....	....	*Mines and Minerals*, Vol. 25 (1905), p. 288.
7	Lead, Trail, B. C.....	6.60	32.21	24.60	2.20	12.60	.....	.....	.....	81.99	.....	....	
8	Rich Lead, Parke's Process	12.56	78.45	4.12	.....	3.00	.88	.....	.....	.....	.....	.....	Original.
9	Lead, Trail, B. C.....	7.10	29.20	30.50	6.10	10.20	.....	.....	.....	.....	.....	.. ..	Original.
10	Lead, Trail, B. C	7.70	31.90	37.60	2.80	12.60	.....	.....	.....	.....	.....	.....	Original.
11	Lead from El Doctor Mine, Mex.....	7.82	2.44	75.34	0 24	12.23	1.95	.....	.....	.....	.....	.....	Original.
12	Copper, Montana Converter Anodes	41.	24.	.....	.....	.....	.....	.....	.....	18.	.....	...	*Trans. Am. Inst. Min. Eng*, 1904, p. 310.
13	Copper, Montana Reverberatory Anodes.	18.	51 4	.....	.....	.....	....	.....	.....	38.	.....	....	
14	Copper, Boston & Montana.....	57.	14.80	2.00	2.60	5.26	5.70	.....	.....	.....	2.0	1.0	Original.
15	Copper, Boston & Montana.....	53.29	12.90	3.30	1.15	trace	1.55	11.96	.....	.....	.....	.....	Original.

butterflies have been chased, for the only questions are, first, whether the solution it is desired to use is cheap enough, safe enough and easily enough contained, and then to see if it will work. What is presented here is, therefore, mainly the result of the elimination process.

Electrolytic slime from lead refining varies in composition, generally much less than would be expected, except in content of bismuth, and analyses are given in Table I.

Electrolytic slime from copper refining is of somewhat different nature. Arsenic, antimony and bismuth stand slightly above copper in the electromotive force series, and below lead by a considerable margin of some 0.3 volt or

soda carbonate in air current, and somewhat dissolved by NaOH in air current. Sodium thiosulphate in air current, sodium tetrathionate and sodium thiosulphate, and sodium sulphide in air current, produce only lead sulphide. Ferric fluosilicate, lead peroxide and fluosilicic acid dissolve lead very easily as led fluosilicate.

Copper very readily dissolves as cupric sulphate by treatment with ferric sulphate. The copper sulphide present in most copper slime is rapidly decomposed, but for good results the temperature should not be high enough to cause the separated sulphur to cohere. Readily changed by HCl in a current of air, and by other chloridizing agents, to cuprous

TABLE II.—SOLUBILITY OF SILVER SULPHATE IN 100° SULPHURIC ACID.

Temperature.	Distilled Water.	5% H_2SO_4 Sol.	10% H_2SO_4 Sol.	20% H_2SO_4 Sol.	30% H_2SO_4 Sol.
20° C	0.774 grams Ag_2SO_4 0 536 " Silver	1.010 grams Ag_2SO_4 0.699 " Silver	0.939 grams Ag_2SO_4 0.650 " Silver	0.658 grams Ag_2SO_4 0 455 " Silver	0.518 grams Ag_2SO_4 .359 " Silver
40° C	0.979 grams Ag_2SO_4 0.678 " Silver	1.637 grams Ag_2SO_4 1.133 " Silver	1.611 grams Ag_2SO_4 1.115 " Silver	1.159 grams Ag_2SO_4 0.802 " Silver	0 931 grams Ag_2SO_4 0.644 " Silver
65° C	1.185 grams Ag_2SO_4 0.820 " Silver	2.884 grams Ag_2SO_4 2.000 " Silver	2.735 grams Ag_2SO_4 1.893 " Silver	2.318 grams Ag_2SO_4 1.605 " Silver	2.072 grams Ag_2SO_4 1.434 " Silver
90° C	1.361 grams Ag_2SO_4 0.942 " Silver	4.040 grams Ag_2SO_4 2.800 " Silver	3.863 grams Ag_2SO_4 2.674 " Silver	3.644 grams Ag_2SO_4 2.523 " Silver	3.705 grams Ag_2SO_4 2.565 " Silver

more, so that in lead slime these metals are found as metals, and in copper slime are present for the most part in the oxidized condition. The sulphur in copper slime comes from using anodes of blister copper, which, according to common theory, contains dissolved sulphurous acid, but more probably cuprous oxide and cuprous sulphide, which have not reacted on each other. On dissolving as anode, the cuprous sulphide goes into the slime, accounting for the high copper content of some slimes. A few analyses are given in Table I.

and cupric chlorides. Ferric fluoride gives readily soluble cupric fluoride. Cupric fluoride and copper do not appear to react to form cuprous fluoride, analogously to cupric chloride and copper. Sodum hypochlorite gives no appreciable action. Sodium thiosulphate in air current makes some copper sulphide.

Ferric fluosilicate, fluosilicic acid and lead peroxide, give readily soluble cupric fluosilicate.

Silver, with ferric sulphate solution, readily dissolves hot

as silver sulphate. The reverse reaction appears to take place on cooling to just the extent that silver sulphate crystallizes out, so that silver sulphate *in crystals* is reducible by ferrous sulphate. When sulphur is present in finely divided form this reaction appears to take place,

$$Ag_2SO_4 + S + 2FeSO_4 = Ag_2S + Fe_2(SO_4)_2$$

For solubility of silver sulphate in sulphuric acid of various strengths see Table 2. Silver in slime is readily changed by chloridizing agents, even by sodium hypochlorite to silver chloride. Hot ferric fluoride gives soluble silver fluoride, as does chromic acid and hydrofluoric acid. Sodium thiosulphate and air produce silver sulphide to some extent. Alkalies and air current give no action, but alkaline sulphides would probably give silver sulphide. Ferric fluosilicate, and lead peroxide and fluosilicic acid, give soluble silver fluosilicate.

Gold in the form in which it occurs in electrolytic slime is only incompletely dissolved with free chlorine and chlorine-generating compounds.

Bismuth is peculiar in that it exists in solution in two different series of salts, which fact appears not to have been previously called attention to. Bismuth and a cold ferric sulphate solution containing free sulphuric acid react very rapidly with almost complete solution. If this solution is heated the bismuth sulphate formed is changed, and basic bismuth sulphate separates. On cooling, the reverse change does not take place. The precipitate is not soluble in sulphuric acid, but if it be washed with soda or ammonia it readily dissolves in cold sulphuric acid. The reaction between bismuth and ferric sulphate takes place slowly on heating, as the metal becomes covered with an insoluble crust of basic sulphate, which protects it. The precipitation of basic bismuth sulphate, on heating, affords a means of precipitating pure bismuth compounds. In the ordinary slime treatment bismuth goes into oxide or basic sulphate. By washing the residue with soda, and extracting with cold 20 per cent sulphuric acid, it can be extracted, but not completely. Chlorine, HCl in air current and other chloridizing agents, readily convert bismuth to chloride. Ferric fluoride gives insoluble bismuth basic fluoride. Ferric fluosilicate and lead peroxide and fluosilicic acid react analogously to ferric sulphate. The combination with bismuth salts of sodium thiosulphate has not been found stable enough to be made use of. Alkalies and sodium sulphide under oxidizing influences will not dissolve bismuth. Sodium hypochlorite has no action.

Antimony is readily converted by ferric sulphate to insoluble trioxide or basic sulphate. This can be readily extracted from the slime with hydrofluoric acid. Antimony trifluoride is entirely dissimilar to antimony trichloride in its reaction with water, not being decomposed by dilution of the solution. It is a very soluble salt; the specific gravity of the saturated solution is nearly 2.5. By electrolysis of antimony trifluoride with lead anodes and cathodes, antimony is deposited, and hydrofluoric acid set free. Chlorine, HCl in air current, ferric chloride, and other chloridizing agents readily form antimony trichloride. Ferric fluoride and antimony pentafluoride readily give antimony trifluoride. Ferric fluosilicate acts analogously to ferric sulphate, but the solution takes up as much as 0.4 per cent of antimony. Addition of tartaric acid to ferric sulphate, even in large amounts, causes the solution of only a small amount of the antimony oxidized. Caustic soda in air current oxidizes antimony somewhat, but dissolves very little, contrary to an incorrect statement by Whitehead.[1] Sodium hypochlorite does not act on antimony. Sodium sulphide in a current of air oxidizes and dissolves antimony rapidly. Sodium carbonate in a current of air causes some oxidation. Sodium thiosulphate in an air current, no action.

Arsenic is the metal most easily dissolved in slime, as it readily dissolves under oxidizing conditions in both acid and alkaline solutions. Sodium hypochlorite dissolves it rapdly. The difficulty with arsenic is not so much to dissolve it, but to dispose of it in solution. It cannot be well electrodeposited, and stands so near antimony, copper and bismuth in the electromotive force series, that it cannot be precipitated by them. Copper can, however, be precipitated in presence of arsenic by electrolysis.

Slime Treatment Processes in Present Use.

Electrolytic copper slime was originally treated, I understand, with nitric acid, which was no doubt good enough when silver in copper was not paid much for, and when it was worth $1.20 per ounce, but the process has been improved, until slime is now generally treated with sulphuric acid and sodium nitrate, which takes out arsenic and copper, and would take out some silver but for the fact that copper plates are hung in the solution to throw dissolved silver out. The current statements that copper slime is treated with sulphuric acid, and air blown through for oxidation are misleading, for the reaction is very slow. Passing sulphurous acid in with the air to oxidize sulphurous acid to sulphuric is worse yet, for the reaction will hardly take place at all.

Air and steam are admitted into the solution containing the slime suspended, for the double purpose of agitation and heating. The copper sulphate produced is usually crystallized, and some plants have apparatus for purifying the solution from arsenic and antimony before crystallizing. The slime is filtered, sometimes with a filter press, and dried in large iron pans placed in or near a flue, or on a large iron floor with a fireplace underneath. Still containing the original antimony, bismuth, lead and sulphur, the slime is melted with soda to slag the antimony, and nitre is always used to oxidize the impurities from the bullion. The doré bullion produced is usually parted with sulphuric acid, and sometimes electrolytically with a silver nitrate bath.

The main objections to this useful process are: Cost of the sulphuric acid and sodium nitrate, and loss of time in completing the reaction. Loss of antimony and bismuth, cost of soda used to slag off antimony, and of nitre for refining. volatilization of antimony, and long time the bullion is in the furnace, both of which cause loss of silver. These objections, while making a serious enough item in the cost of refining a ton of copper producing 15 pounds of slime, make a cost altogether too high per ton of lead, producing 60 to 100, or even 300 pounds of slime per ton.

Electrolytic Slime Treatment.

The improvements brought about by electrolytic treatment are regeneration of the sulphuric acid used as solvent, and of the oxidizing agent at the same time, at a much lower cost than for new chemicals, recovery of the copper as electrolytic copper, and incidentally converting outside copper oxide into electrolytic metal, and recovery of the arsenic as arsenious acid if desired, and practically complete recovery of antimony and bismuth as pure metal. Simple melting of a smaller quantity of material, with a smaller amount of soda and no oxidizing agent, in closed crucibles or retorts in much less time and without loss by volatilization, the volatile antimony having been removed. Electrolytic parting of the silver, gold, bismuth, with traces of copper, antimony arsenic, by an improved method at less cost, and with better results.

A scheme of the process is given as Fig. 1.

The ferric sulphate solution as it comes from the electrolytic tanks contains 4-5 per cent of iron as ferric sulphate, 0.5-1 per cent as ferrous sulphate, 1 per cent copper as cupric sulphate, and from 4-6 per cent of free sulphuric acid. The solution is agitated in a lead-lined tank, by means of a jet of air and steam in the usual manner, and copper oxide, copper scale or similar material added, which dissolves rapidly, any cuprous oxide being entirely brought into solution be-

[1] Mines and Minerals, Vol. 25, 1905, Jan., p. 286.

cause of the presence of ferric sulphate. The slime to be treated is then added, when these reactions take place quickly:

$Cu + Fe_2(SO_4)_3 = CuSO_4 + 2FeSO_4$

$Pb + Fe_2(SO_4)_3 = PbSO_4 + 2FeSO_4$

$2As + 3H_2O + 3Fe_2(SO_4)_3 = As_2O_3 + 6FeSO_4 + 3H_2SO_4$

$2Sb + 3H_2O + 3Fe_2(SO_4)_3 = Sb_2O_3 + 6FeSO_4 + 3H_2SO_4$

$2Bi + 3H_2O + 3Fe_2(SO_4)_3 = Bi_2O_3 + 6FeSO_4 + 3H_2SO_4$

$2Ag + Fe_2(SO_4)_3 = Ag_2SO_4 + 2FeSO_4$

It will be noticed from the reactions that acid is set free in the oxidation of arsenic, antimony and bismuth. This acid will be neutralized in the same way with copper oxide just before the next treatment, in order that the process may

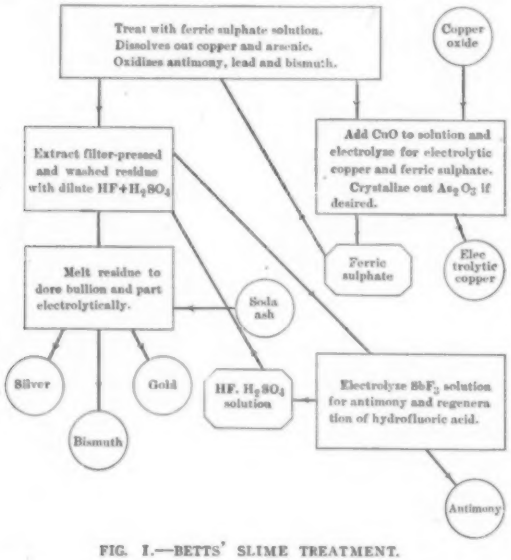

Treat with ferric sulphate solution. Dissolves out copper and arsenic. Oxidizes antimony, lead and bismuth.
Copper oxide
Extract filter-pressed and washed residue with dilute HF+H2SO4
Add CuO to solution and electrolyze for electrolytic copper and ferric sulphate. Crystallize out As2O3 if desired.
Ferric sulphate
Electrolytic copper
Melt residue to dore bullion and part electrolytically.
Soda ash
Silver
Gold
Bismuth
HF. H2SO4 solution
Electrolyze SbF5 solution for antimony and regeneration of hydrofluoric acid.
Antimony

FIG. 1.—BETTS' SLIME TREATMENT.

be regenerative. Some silver is almost sure to dissolve, even if an insufficient quantity of ferric sulphate is used, and this is precipitated before filtration by hanging copper plates in the solution. Several hours are necessary to throw the silver out. Silver can be largely dissolved, before filtration, and later precipitated separately by copper, but the precipitate is very voluminous, and the silver contains antimony and arsenic oxides, so that it is found to be cheaper to leave the silver in the slime, although the opportunity is present to remove most of it.

For 100 pounds of electrolytic lead slime, containing perhaps 5 pounds of arsenic as a fair average, about 22 cubic feet of solution containing 4 per cent iron as ferric sulphate are used. The amount of arsenic after the first treatment rises to about .35 per cent.

The residue must be washed free from copper and iron solution, in order that the subsequent antimony extraction may give a pure deposit of antimony, which can be best done with a filter press. It does not pay to evaporate the wash waters, because ferrous sulphate is a very cheap salt, worth from 3 to 5 dollars per ton, and the wash water is accordingly run to waste over scrap iron to precipitate out copper. If 10 per cent of the solution is run to waste daily in this way, arsenic will eventually stand in the solution at from 1 to 3 per cent, which is not too high for the deposition of pure copper, under the conditions of temperature and current density used.

Arsenious acid is soluble in about ten parts of the hot solution and 100 parts of cold, so that we have here the means of saving the arsenic by crystallization from the solution, if it became necessary with special slimes, or if the arsenic should be worth saving.

The amount of bismuth and antimony dissolved is small, and they do not deposit with the copper. The reduced iron and copper sulphate solution, containing 3 to 4 per cent of copper, is then electrolyzed for copper and ferric sulphate, as described below.

The filter-press cakes of silver, lead sulphate, bismuth and antimony oxides and gold are then lixiviated in a lead-lined tank, similar to the one used for copper dissolving, with air agitation, in a solution of 2 to 4 per cent sulphuric acid, and 12 or 15 per cent hydrofluoric acid. The antimony dissolves in the course of 4 or 5 hours to the extent of some 90 per cent of the total. If the slime was previously well washed pure electrolytic antimony will be subsequently obtained.

The resulting insoluble portion, after filtration of the antimony on a cloth filter, can be dried and mixed with one-fourth to one-third of its weight of soda ash; that is, 7 to 10 pounds per ton of lead, and melted down to doré bullion, containing the bismuth of the slime. This is parted electrolytically, as described later in this article.

For copper slimes containing sulphur and little antimony it is preferable to extract the sulphur, antimony, selenium and tellurium by caustic soda solution, which reacts with the sulphur to form sodium sulphide, dissolving antimony as sulphantimonite.

Other methods of oxidation of the ferric sulphate were tried, as it was feared at first that the electrolysis of the copper iron sulphates for copper and ferric sulphate could not be practically carried out. For example, allowing the solution to trickle down cloths in the presence of air, crystallization of ferrous sulphate and calcination to basic ferric

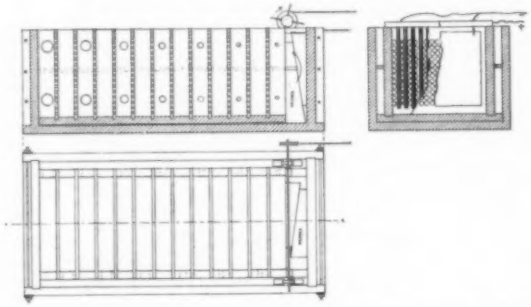


FIG. 2.—PLANT FOR DEPOSITING COPPER AND REGENERATING FERRIC SULPHATE.

sulphate and resolution of the same were found to be far inferior to electrolysis.

ELECTROLYTIC RECOVERY OF COPPER AND FERRIC SULPHATE.

After preliminary experiments a tank was constructed as shown in Fig. 2. The tank is intended to be lined with lead, but this was omitted in this tank, owing to the experimental nature of the work. The diaphragms are of cypress, five-eighth inch thick, perforated with five-eighth inch holes as closely as possible. The holes are closed with asbestos wads, put in wet, which do not drop out. The boards should be wetted before the asbestos wads are put in, otherwise the unequal expansion will split them. Arc-light carbons were used for anodes. One of the side channels in the tank contains the anolyte and the other the catholyte. Circulation is kept up by wooden paddle wheels, the solution flowing through the upper holes into the several compartments, and out again through the lower ones. The proper circulation is insured by boards placed longitudinally in the side channels, but not reaching quite to the ends beneath the paddle wheels. The sills and uprights of the distance frames are in separate pieces, and the whole is held together by wedges at one end. The spaces not closed by the wedges at one side are closed up with asbestos to prevent mixing of the anolyte and catholyte. The solution is fed to the cathode compartments, and siphoned off from the anode compartments, so that a slight

difference in level exists between the anolyte and catholyte, which causes just sufficient flow through the asbestos.

The tank worked well for a few minutes, but soon polarization was set up, gassing occurred at the anodes, and they were soon covered with a slimy layer. Everything that was thought could help was tried, and finally I mounted the anodes on a frame resting on rollers, and gave them a reciprocating motion through the electrolyte. After the anodes were cleaned they remained clean, gassing stopped, the electromotive force dropped, and the efficiency went up to 100 per cent and remained there. The result was surprisingly good and even arc-light carbons will do for the work, but Acheson anodes are better. It is often stated that ordinary carbon anodes are good enough for electrolyses evolving chlorine at the anode, but that carbon will not stand in sulphate solutions. To this I do not agree, for the question is simply one of electromotive force, and not the particular nature of the anion liberated. The difference in voltage of a moving and stationary carbon electrode in ferrous sulphate is over one-half volt.

Deposition of Antimony.

Electrolysis of antimony trifluoride for antimony is carried out in a lead-lined tank, with sheet lead cathodes and lead rods as anodes, with a cathode current density of 10 to 20 amperes per square foot, and twice that at the anode, with an electromotive force of from 2.6 to 2.8 volts. There is a considerable tendency to form antimony pentafluoride at the anodes, which oxidized the cathode deposit. The current efficiency would be low were it not for a protecting layer of cotton cloth around each anode rod, by which means, and by the higher anode density, a current efficiency of 92 or 95 per cent is easily obtained. The antimony deposit is dense and solid, without projecting trees, and does not contain combined antimony salts, as some antimony deposits are said to do. Antimony contains about 1 per cent bismuth.

Parting.

Electrolytic parting is now carried on with a nitrate of silver bath, by either of two processes known as Moebius' and Balbach's. They are unquestionably good processes, but they have the limitation imposed by the lack of large quantity of free acid in the solution, or requiring more power than would otherwise be necessary, and are not suitable for handling anodes containing bismuth. It is also said that lead in the anodes goes into lead peroxide, which would not, however, appear probable from an electrochemical point of view. For parting, I use the silver salt of a non-oxidizing acid, with a large excess of the acid, the best results being obtained with methyl-sulphuric acid. The bismuth of the anodes easily goes into solution along the traces of antimony present, and the lead, copper and silver, and may accumulate to the extent of 4 or 5 per cent without precipitating basic salt. The silver deposit, particularly when 1 part of gum arabic or gelatine is present for every 12,000 to 15,000 parts of solution, is quite solid, and entirely free from projections. Electrolysis may be carried on for forty-eight hours easily without any danger of short-circuits from the growth of projections. It is found rather an advantage than otherwise that the silver deposit is not quite solid, for it can be scraped from the sheet-silver cathodes used, and they can be used over and over again, instead of melting them down with the deposit, as is done with copper and lead cathodes.

The copper and bismuth of the anodes accumulate in the solution at the expense of silver deposited on the cathodes. After silver is reduced to about 1.5 per cent, the solution should be replaced. For working up the solution and saving the bismuth, the remaining silver is first precipitated by metallic copper, and then copper and bismuth by metallic lead, and the lead methyl-sulphate decomposed by silver sulphate, giving lead sulphate and solution of silver methyl-sulphate to be used over again. The precipitate of copper and bismuth can be treated with ferric sulphate to remove the copper and the bismuth oxide can be melted to metal.

Cost of Treating Slime Electrolytically.

For a particular case, on the basis of treatment of 1 ton of lead bullion, containing antimony 1 per cent, silver 1 per cent, arsenic ¼ per cent, bismuth .2 per cent, copper ½ per cent, and 1 ounce of gold per ton, the cost is as follows:

Chemicals, as follows			$0.33
Ferrous sulphate, 5 per cent daily loss 24 pounds, at $5 per ton	6	cents	
Sulphuric acid for copper-iron solution, antimony solution, making silver sulphate, 9 pounds at 1 cent	9	"	
Soda ash, 10 pounds, at 1 cent	10	"	
Other chemicals, nitric acid for gold, losses in silver solution	5	"	
Hydrofluoric acid loss 1 pound, at 3 cents	3	"	
	33	"	
Power			
for depositing 20 pounds antimony	18	kw.-hrs.	
for depositing 52 pounds copper and making ferric sulphate	25	"	
Refining 20 pounds silver	1	kw.-hr.	
Total, at $30 per hp.-year of 365 days	44	kw.-hrs.	$0.20
2250 pounds steam for heating solution			0.05
20 pounds coal for melting antimony and silver, at $4			0.04
Labor, 10 men required for plant refining slime from 100-ton lead refinery			.25
Allow for supplies and repairs			.10
Total cost per ton of lead of analysis given			$0.97

The copper oxide used is not figured in as an element of cost, as copper in oxide is worth less than as electrolytic metal, so that a gain is made, rather than the cost increased.

This works out at a cost of about one and one-half cents per pound of slime, and when it is remembered that it costs copper refineries one cent per ounce, or more, to collect their silver—that is, with a slime carrying 50 per cent silver, seven cents per pound, or more, to treat it, besides throwing away the by-products and losing at least one per cent of silver, which loss may be apparently more than covered by the loss in commercial fire-assay—the superiority of the electrochemical method, working at less than one-fourth the cost, and saving everything of value, is apparent. The electrochemical process on a 100-pound lot of slime has given a return of silver equal to corrected fire-assay, or one and one-fourth per cent higher than commercial assay.

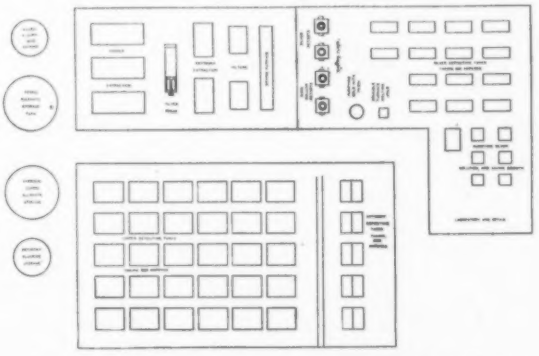


FIG. 3.—ELECTROLYTIC SLIME ARRANGEMENT OF PLANT

The general arrangement of the plant is shown in Fig. 3.

My thanks are due Dr. Edward F. Kern for much valuable assistance.

Metallurgical Calculations.—II.

By J. W. Richards, Ph. D.
Professor of Metallurgy in Lehigh University.

The Application of Thermochemical Principles.

The ordinary interpretation of the chemical equation by weight gives us the quantitative relations governing the reactions of substances upon each other, when the reaction proceeds to a finish. Unfortunately, much chemical instruction, as given in our elementary schools, and even in some of the higher ones, stops with the consideration of the weight relations and does not proceed to those equally important relations, the energetics of chemical reactions. In most of the reactions which occur in practical metallurgy, the quantity of the combustible used, or, more broadly, the amount of energy in the form of heat or electrical energy, necessary for producing the reactions desired, is the controlling factor regulating the practicability or impracticability, the commercial success or failure, of the process.

The relative values of fuels, the manufacture and utilization of gas, the principles of the regenerative furnace, the Bessemer process, electric reduction, and a host of metallurgical processes, depend essentially on the realization and utilization of chemical energy, and the only way to become conversant with the amounts of energy involved or evolved in these operations is to understand thoroughly the thermochemistry of the reactions concerned.

Thermochemical Nomenclature.

The heat evolved when compounds are formed from the elements (in a few cases heat is absorbed) is determined experimentally by the use of the calorimeter. This branch is sometimes called "chemical calorimetry;" it is practically a department of experimental physics. The data obtained give the heat evolved for the total change from the components at room temperature to the resulting products, at room temperature (or very near to it), and may be expressed per unit of weight of either component or of the substance formed. Thus, if carbon is burned in a calorimeter to carbonic acid gas, CO^2, the heat evolved may be reported or expressed as

8100 gram calories per gram of carbon burnt.

Or, 3037 gram calories per gram of oxygen used.

Or, 2209 gram calories per gram of CO^2 formed.

Of these three methods of expressing the results, the first is the more often used, especially by the physicist.

The chemist, however, finds it often more convenient and logical to express these heats of combination per formula weight of the substances combining and of product formed. *E. g.*: In the case of CO^2, which contains 12 parts of carbon and 32 of oxygen in 44 of the gas, the chemist would write,

$$(C, O^2) = 97{,}200,$$

meaning thereby that when 12 grams of carbon is burnt by 32 grams of oxygen, forming 44 grams of carbon dioxide, there is evolved 97,200 gram-calories. These are the laboratory units; for practical purposes, we call the weights kilograms and the heat units kilogram-calories (gram-calories are abbreviated to "cal."; kilogram-calories to "Cal."). Ostwald, in his thermochemical tables writes $(C, O^2) = 972$ K, where K represents a unit 100 times as large as a gram-calorie, if weights are taken as being grams. Berthelot writes $(C, O^2) = 97.2$, where the heat units are kilogram-calories, if the weights concerned are taken as grams. Both these methods of expression are liable to cause confusion; the writer prefers to follow the older thermochemists (Hess, Naumann) and to use the larger number, *e. g.*, 97,200, which then means gram-calories, if weights are taken in grams (laboratory units), and kilogram-calories, if weights are taken in kilograms (practical units).

If it is desired to work in "British Thermal Units" (1 B. T. U. is the heat needed to raise one pound of water one degree Fahrenheit), the weights represented by the formula may be called pounds, and then the expression is as follows:

$$(C, O^2) = [97{,}200 \times 9/5] = 174{,}960 \text{ B. T. U.}$$

The factor 9/5 is simply the relation of 1° C. to 1° F.; the reasoning is, of course, that if the combustion of 12 kilograms of carbon evolves 97,200 kilogram calories, or would heat 97,200 kilograms of water 1° C., that the combustion of 12 pounds of carbon would heat 97,200 pounds of water 1° C., or 174,960 pounds 1° F. From the equation as thus expressed and interpreted, the heat of combination per pound of carbon burnt, or of oxygen used, or of product formed, may be found in B. T. U. by dividing 174,960 by 12, 16 or 44, respectively.

Since it is very inconvenient, as well as unscientific, to have the two unrelated heat units, with their resulting double sets of experimental data, I strongly recommend the use of the metric data and metric-Centigrade heat unit. It is, however, sometimes convenient, when all the data of a problem are given in English weights, to use as the unit of heat the "pound — 1° C.," or the heat required to raise the temperature of one pound of water 1° C. This may be called the "pound cal.," as distinguished from the B. T. U. The advantage of using it is that all the experimental data of the metric system units are at once transferable to the English weights. *E. g.*, $(C, O^2) = 97{,}200$ pound cal., if the weights concerned in the formula (12, 32, 44) are called pounds.

The thermochemist gives all his experimental data in the form above explained, and we will now give all the important thermochemical data known which are useful in metallurgical calculations, *the data being for the reactions beginning and ending at* 15° C. (60° F.):

Thermochemical Data.

Heat of Formation of Oxides.

Formula.	*Molecular Weights.*	*Molecular Heat of Formation.*	*In dilute Solution.*
(Mg, O)	$24 + 16 = 40$	143,400	148,800
(Ba, O)	$137 + 16 = 153$	133,400	161,500
(Ca, O)	$40 + 16 = 56$	131,500	149,600
(Sr, O)	$87 + 16 = 103$	131,200	158,400
(Al^2, O^3)	$54 + 48 = 102$	392,600	.....
(Na^2, O)	$46 + 16 = 62$	100,900	155,900
(K^2, O)	$78 + 16 = 94$	98,200	165,200
(Si, O^2)	$28 + 32 = 60$	180,000	.....
(Mn, O)	$55 + 16 = 71$	90,900	.....
(B^2, O^3)	$22 + 48 = 70$	272,600	279,900
(Zn, O)	$65 + 16 = 81$	84,800	.....
(Mn^3, O^4)	$165 + 64 = 229$	328,000	.....
(P^2, O^5)	$62 + 80 = 142$	365,300	.....
(Sn, O)	$118 + 16 = 134$	70,700	.....
(Sn, O^2)	$118 + 32 = 150$	141,300	.....
(CO, O)	$28 + 16 = 44$	68,040	73,940
(H^2, O)	$2 + 16 = 18$	70,400 (solid) 69,000 (liquid) 58,060 (gas)	....
(Fe^3, O^4)	$168 + 64 = 232$	270,800	.....
(Cd, O)	$112 + 16 = 128$	66,300	.....
(Fe, O)	$56 + 16 = 72$	65,700	.....
(Fe^2, O^3)	$112 + 48 = 160$	195,600	.....
(Co, O)	$59 + 16 = 75$	64,100	.....
(Mn, O^2)	$55 + 32 = 87$	125,300	.....
(Ni, O)	$58.5 + 16 = 74.5$	61,500	.....
(Sb^2, O^3)	$240 + 48 = 288$	166,900	.....
(As^2, O^3)	$150 + 48 = 198$	156,400	148,900
(Pb, O)	$207 + 16 = 223$	50,800	.....
(C, O^2)	$12 + 32 = 44$	97,200 (gas)	103,100
(Bi^2, O^3)	$416 + 48 = 464$	139,200	.....
(Sb^2, O^5)	$240 + 80 = 320$	231,200	.....
(As^2, O^5)	$150 + 80 = 230$	219,400	225,400
(Cu^2, O)	$127.2 + 16 = 143.2$	43,800	.....

Formula.	*Molecular Weights.*	*Molecular Heat of Formation.*	*In dilute Solution.*
(Tl^2, O)	408 + 16 = 424	42,800	39,700
(Cu, O)	63.6 + 16 = 79.6	37,700	.....
(Ba, O^2)	137 + 32 = 169	145,500	.....
(S, O^3)	32 + 32 = 64	69,260 (gas)	77,600
(Pb, O^2)	207 + 32 = 239	63,400	.....
(S. O^2)	32 + 48 = 80	91,900	141,000
(Tl^2, O^3)	408 + 48 = 456	87,600	.....
(C, O)	12 + 16 = 28	29,160 (gas)	.....
(Hg^2, O)	400 + 16 = 416	22,200	.....
(Hg, O)	200 + 16 = 216	21,500	.....
(Te, O^3)	125.5 + 32 = 157.5	.....	78,300
(Pd, O)	106 + 16 = 122	21,000	.....
(Pt, O)	195 + 16 = 211	17,000 (?)	.....
(Ag^2, O)	216 + 16 = 232	7,000	.....
(Au^2, O^3)	394 + 48 = 442	−11,500	.....
Heat of Formation of Hydrates.			
(Li, O, H)	7 + 16 + 1 = 24	112,300	118,100
(Mg, O^2, H^2)	24 + 32 + 2 = 58	217,800	.....
(Sr, O^2, H^2)	87 + 32 + 2 = 121	217,300	227,400
(Ca, O^2, H^2)	40 + 32 + 2 = 74	215,600	219,500
(K, O, H)	39 + 16 + 1 = 56	104,600	117,100
(Na, O, H)	23 + 16 + 1 = 40	102,700	112,500
(N, H^5, O)	14 + 5 + 16 = 35	88,800	90,000
(Al^2, O^3, H^3)	27 + 48 + 3 = 78	301,300	.....
(H, O, H)	1 + 16 + 1 = 18	70,400 (solid) 69,000 (liquid) 58,060 (gas)	....
(Tl, O, H)	204 + 16 + 1 = 221	57,400	54,300
(Bi, O^3, H^3)	208 + 48 + 3 = 259	171,700	.....
(Zn, O^2, H^2)	65 + 32 + 2 = 99	83,500	.....
(Te, O^3, H^2)	127 + 32 + 2 = 161	78,300	.....
(Te, O^3, H^3)	127 + 48 + 3 = 178	.....	99,500
(Se, O^3, H^2)	79 + 32 + 2 = 113	52,400	51,500
(Se, O^4, H^2)	79 + 48 + 3 = 130	.....	79,300
(Tl, O^3, H^3)	204 + 48 + 3 = 255	43,800	.....
Heat of Formation of Sulphides.			
(Li^2, S)	14 + 32 = 46	.....	115,400
(K^2, S)	78 + 32 = 110	103,500	113,500
(Ba, S)	137 + 32 = 169	102,900	109,800
(Sr, S)	87 + 32 = 119	99,300	106,700
(Ca, S)	40 + 32 = 72	94,300	100,600
(Na^2, S)	46 + 32 = 78	89,300	104,300
(Mg, S)	24 + 32 = 56	79,400	.....
(K, S^2)	39 + 64 = 103	59,300	59,700
(Na, S^2)	23 + 64 = 87	49,500	54,400
(Mn, S)	55 + 32 = 87	45,600	.....
(Zn, S)	65 + 32 = 97	43,000	.....
(Al^2, S^3)	54 + 96 = 150	126,400	.....
(N, H^5, S)	14 + 5 + 32 = 51	40,000	36,700
(Cd, S)	112 + 32 = 144	34,400	.....
(B^2, S^3)	22 + 96 = 118	75,800	.....
(Fe, S)	56 + 32 = 88	24,000	.....
(Co, S)	59 + 32 = 91	21,900	.....
(Tl^2, S)	204 + 32 = 236	21,600	.....
(Cu^2, S)	127.2 + 32 = 159.2	20,300	.....
(Pb, S)	207 + 32 = 239	20,200	.....
(Si, S^2)	28 + 64 = 92	40,000	.....
(Ni, S)	58.5 + 32 = 90.5	19,500	.....
(Sb^2, S^3)	240 + 96 = 336	34,400	.....
(Hg, S)	200 + 32 = 232	10,600	.....
(Cu, S)	63.6 + 32 = 95.6	10,100	.....
(H^2, S)	2 + 32 = 34	4,800 (gas)	9,500
(Ag^2, S)	216 + 32 = 248	3,000	.....
(C, S^2)	12 + 64 = 76	−25,400 (gas) −19,000 (liquid)	....
(I, S)	127 + 32 = 159	0,000	.....

Formula.	*Molecular Weights.*	*Molecular Heat of Formation.*	*In dilute Solution.*
Heat of Formation of Selenides.			
(Li^2, Se)	14 + 79 = 93	83,000	93,700
(K^2, Se)	78 + 79 = 157	79,600	87,900
(Ba, Se)	137 + 79 = 216	69,900	.....
(Sr, Se)	87 + 79 = 166	67,600	.....
(Na^2, Se)	46 + 79 = 125	60,900	78,600
(Ca, Se)	40 + 79 = 119	58,000	.....
(Zn, Se)	65 + 79 = 144	30,300	.....
(Cd, Se)	112 + 79 = 191	23,700	.....
(Mn, Se)	55 + 79 = 134	22,400	.....
(N, H^4, Se)	14+5 + 79 = 98	17,800	12,800
(Cu, Se)	63.6 + 79 = 142.6	17,300	.....
(Pb, Se)	207 + 79 = 286	17,000	.....
(Fe, Se)	56 + 79 = 135	15,200	.....
(Ni, Se)	58.5 + 79 = 137.5	14,700	.....
(Co, Se)	59 + 79 = 138	13,900	.....
(Tl^2, Se)	408 + 79 = 487	13,400	.....
(Cu^2, Se)	127.2 + 79 = 206.2	8,000	.....
(Hg, Se)	200 + 79 = 279	6,300	.....
(Ag^2, Se)	216 + 79 = 295	2,000	.....
(H^2, Se)	2 + 79 = 81	−25,100 (gas)	−15,800
(N, Se)	14 + 79 = 93	−42,300	.....
Heat of Formation of Tellurides.			
(Zn, Te)	65 + 126 = 191	31,000	.....
(Cd, Te)	112 + 126 = 138	16,600	.....
(Co, Te)	59 + 126 = 185	13,000	.....
(Fe, Te)	56 + 126 = 182	12,000	.....
(Ni, Te)	58.5 + 126 = 184.5	11,600	.....
(Tl^2, Te)	408 + 126 = 534	10,600	.....
(Cu^2, Te)	127.2 + 126 = 253.2	8,200	.....
(Pb, Te)	207 + 126 = 333	6,200	.....
(H^2, Te)	2 + 126 = 128	−34,900 (gas)	.....
Heat of Formation of Arsenides.			
(H^3, As)	3 + 75 = 78	−44,200 (gas)	.....
Heat of Formation of Antimonides.			
(H^3, Sb)	3 + 120 = 123	−86,800 (gas)	.....
Heat of Formation of Phosphides.			
(Mn^3, P^2)	165 + 62 = 227	70,900	.....
(H^3, P)	3 + 31 = 34	4,900 (gas)	.....
(Fe, P)	56 + 39 = 95	nearly zero.	.....
Heat of Formation of Nitrides.			
(H^3, N)	3 + 14 = 17	12,200 (gas) 16,600 (liquid)	21,000 .....
(Ba^3, N^2)	411 + 28 = 439	149,400	.....
(Li^3, N)	21 + 14 = 35	49,500	.....
(K, H^3, N)	39 + 3 + 14 = 56	30,700	.....
Heat of Formation of Metallic Hydrides.			
(Sr, H^2)	87 + 2 = 89	38,400	.....
(Ba, H^2)	137 + 2 = 139	37,500	.....
(Pt^{10}, H)	1,950 + 1 = 1,951	14,200	.....
(Pd^{12}, H)	1,590 + 1 = 1,591	4,600	.....
(Si, H^4)	28 + 4 = 32	−6,700 (gas)	.....
Heat of Formation of Carbides.			
(Al^4, C^3)	108 + 36 = 144	232,000	.....
(Mn, C^2)	55 + 24 = 79	114,400 (Ponthiere)	..
(Mn, C^3)	55 + 36 = 91	9,900 (Berthelot)	..
(Fe^3, C)	168 + 12 = 180	8,460	.....
(Ca, C^2)	40 + 24 = 64	−6,250	.....
(Na, C)	23 + 12 = 35	−4,400	.....
(Li, C)	7 + 12 = 19	−5,750	.....
(N^2, C^2)	28 + 24 = 52	−73,000 (gas)	−67,100
(Ag, C)	108 + 12 = 120	−43,575	.....

Heat of Formation of Silicides.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Solution.
(Mn^3, Si^2)	385 + 56 = 441	47,400	.....
(H^4, Si)	0 + 28 = 32	— 6,700 (gas)	.....

Heat of Formation of Fluorides.

(Sr, F^2)	87 + 38 = 125	224,020	.....
(Ba, F^2)	137 + 38 = 175	224,000	221,500
(Li, F)	7 + 19 = 26	.....	116,880
(K, F)	39 + 19 = 58	110,000	113,600
(Ca, F^2)	40 + 38 = 78	216,450	.....
(Mg, F^2)	24 + 38 = 62	209,500	.....
(Na, F)	23 + 19 = 42	109,720	109,120
(N, H^4, F)	14 + 4 + 19 = 37	101,250	99,750
(Al, F^3)	27 + 57 = 84	.....	275,220
(B, F^3)	11 + 57 = 68	.....	219,345
(Mn, F^2)	55 + 38 = 93	.....	153,310
(Zn, F^2)	65 + 38 = 103	.....	138,220
(Si, F^4)	28 + 76 = 104	275,920 (gas)	.....
(Fe, F^2)	56 + 38 = 94	.....	125,220
(Cd, F^2)	112 + 38 = 150	.....	121,720
(Co, F^2)	59 + 38 = 97	.....	120,340
(Ni, F^2)	58.5 + 38 = 96.5	.....	118,980
(Fe, F^3)	56 + 57 = 113	.....	164,940
(Tl, F)	204 + 19 = 223	.....	54,405
(Pb, F^2)	207 + 38 = 245	101,600	.....
(H, F)	1 + 19 = 20	38,500 (gas)	50,300
(Sb, F^3)	120 + 57 = 177	.....	136,680
(Cu, F^2)	63.6 + 38 = 101.6	.....	88,160
(Ag, F)	108 + 19 = 127	22,070	25,470

Heat of Formation of Chlorides.

(K, Cl)	39 + 35.5 = 74.5	105,700	101,200
(Ba, Cl^2)	137 + 70 = 207	197,100	198,300
(Na, Cl)	23 + 35.5 = 58.5	97,900	96,600
(Li, Cl)	7 + 35.5 = 42.5	93,900	102,300
(Sr, Cl^2)	87 + 71 = 158	184,700	195,850
(Ca, Cl^2)	40 + 71 = 111	169,900	187,400
(N, H^4, Cl)	14 + 4 + 35.5 = 63.5	76,800	72,800
(Mg, Cl^2)	24 + 71 = 95	151,200	187,100
(Al, Cl^3)	27 + 106.5 = 133.5	161,800	238,100
(Mn, Cl^2)	55 + 71 = 126	112,000	128,000
(Zn, Cl^2)	65 + 71 = 136	97,400	113,000
(Tl, Cl)	204 + 35.5 = 239.5	48,600	38,400
(Cd, Cl^2)	112 + 71 = 183	93,700	96,400
(Pb, Cl^2)	207 + 71 = 278	83,900	77,900
(Fe, Cl^2)	56 + 71 = 127	82,200	100,100
(Sn, Cl^2)	118 + 71 = 189	80,900	.....
(Co, Cl^2)	59 + 71 = 130	76,700	95,000
(Ni, Cl^2)	58.5 + 71 = 129.5	74,700	93,900
(Cu, Cl)	63.5 + 35.5 = 99	35,400	.....
(Sn, Cl^4)	118 + 142 — 260	129,800 (liquid)	.....
(Fe, Cl^3)	56 + 106.5 = 162.5	96,150	127,850
(Hg, Cl)	200 + 35.5 = 235.5	31,320	.....
(Sb, Cl^3)	120 + 106.5 = 226.5	91,400	.....
(Bi, Cl^3)	208 + 106.5 = 314.5	90,800	.....
(B, Cl^3)	11 + 106.5 = 117.5	89,100 (gas)	.....
(Ag, Cl)	108 + 35.5 = 143.5	29,000	.....
(Hg, Cl^2)	200 + 71 = 271	53,300	50,300
(Cu, Cl^2)	63.6 + 71 = 134.6	51,400	62,500
(As, Cl^3)	75 + 106.5 = 181.5	71,500	.....
(H, Cl)	1 + 35.5 = 36.5	22,000	39,400
(Sb, Cl^5)	120 + 177.5 = 297.5	104,500 (liquid)	.....
(Pd, Cl^2)	106 + 71 = 177	40,500	.....
(Pt, Cl^4)	195 + 142 = 337	60,200	79,800
(Au, Cl^2)	197 + 106.5 = 303.5	22,800	27,200
(Au, Cl)	197 + 35.5 = 232.5	5,800	.....

Heat of Formation of Hydrocarbons.

(All formed in state of gas, unless otherwise specified.)

Name.	Formula.	Molecular Weights.	Molecular Heat of Formation.
Methane (marsh gas)	(C, H^4)	12 + 4 = 16	22,250
Ethane (ethylene hydride)	(C^2, H^6)	24 + 6 = 30	26,650
Propane (propylene hydride)	(C^3, H^8)	36 + 8 = 44	33,850
Ethylene (olefiant gas)	(C^2, H^4)	24 + 4 = 28	— 11,250
Propylene	(C^3, H^6)	36 + 6 = 42	— 6,050
Toluene	(C^7, H^8)	84 + 8 = 92	5,650 (liquid)
Benzene	(C^6, H^6)	72 + 6 = 78	— 750 (liquid) — 7,950 (gas)
Turpentine	(C^{10}, H^{16})	120 + 16 = 136	7,550 (liquid) — 1,850 (gas)
Naphthaline	(C^{10}, H^8)	120 + 8 = 128	— 19,450 (solid) — 24,050 (liquid)
Anthracene	(C^{14}, H^{10})	168 + 10 = 178	— 39,050 (solid)
Acetylene	(C^2, H^2)	24 + 2 = 26	— 54,750
Methyl-alcohol (wood spirit)	(C, H^4, O)	12 + 4 + 16 = 32	65,050 (liquid) 56,650 (gas)
Ethyl-alcohol (alcohol)	(C^2, H^6, O)	24 + 6 + 16 = 46	73,250 (liquid) 63,150 (gas)
Acetone	(C^3, H^6, O)	36 + 6 + 16 = 58	69,650 (liquid) 62,150 (gas)

Heat of Formation of Carbonates.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Solution.
(Ba, C, O^3)	137 + 12 + 48 = 197	286,300	.....
(K^2, C, O^3)	78 + 12 + 48 = 138	282,100	288,600
(Sr, C, O^3)	87 + 12 + 48 = 147	281,400	.....
(Ca, C, O^3)	40 + 12 + 48 = 100	273,850	.....
(Na^2, C, O^3)	46 + 12 + 48 = 106	273,700	279,300
(Mg, C, O^3)	24 + 12 + 48 = 88	269,900	.....
(Mn, C, O^3)	55 + 12 + 48 = 115	210,300	.....
(N, H^4, C, O^3)	14 + 5 + 12 + 48 = 79	208,600	202,300
(Zn, C, O^3)	65 + 12 + 48 = 125	197,500	.....
(Fe, C, O^3)	56 + 12 + 48 = 116	187,800	.....
(Cd, C, O^3)	112 + 12 + 48 = 172	183,200	.....
(Pb, C, O^3)	207 + 12 + 48 = 267	170,000	.....
(Cu, C, O^3)	63.6 + 12 + 48 = 123.6	146,100	.....
(Ag^2, C, O^3)	216 + 12 + 48 = 276	123,800	.....

Heat of Formation of Bicarbonates.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Solution.
($KHCO_3$)	39 + 1 + 12 + 48 = 100	233,300	228,000
($NaHCO_3$)	23 + 1 + 12 + 48 = 84	227,000	222,700

Heat of Formation of Nitrates.

(KNO_3)	39 + 14 + 48 = 101	119,000	110,700
($NaNO_3$)	23 + 14 + 48 = 85	110,700	106,000
($Zn(NO_3)_2$)	65 + 28 + 96 = 187	.....	131,700
($Pb(NO_3)_2$)	207 + 28 + 96 = 331	105,400	98,200
($Cu(NO_3)_2$)	63.5 + 28 + 96 = 187.5	.....	81,300
(HNO_3)	1 + 14 + 48 = 63	34,400 gas	48,800
($AgNO_3$)	108 + 14 + 48 = 170	28,700	23,000

Heat of Formation of Phosphates.

($Ca_3(PO_4)_2$)	120 + 62 + 128 = 310	919,200	.....
($Mg_3(PO_4)_2$)	72 + 62 + 128 = 262	910,600	.....
(Na_3PO_4)	69 + 31 + 64 = 164	452,400	.....

Heat of Formation of Silicates.

($CaSiO_3$)	40 + 28 + 48 = 116	344,400	.....
($MnSiO_3$)	55 + 28 + 48 = 131	276,300	.....
($FeSiO_3$)	56 + 28 + 48 = 130	254,600	.....

Heat of Formation of Sulphates.

(K_2SO_4)	78 + 32 + 64 = 174	344,300	337,700
($BaSO_4$)	137 + 32 + 64 = 233	339,400	.....
(Li_2SO_4)	14 + 32 + 64 = 110	333,500	339,600
($SrSO_4$)	87 + 32 + 64 = 183	330,200	.....
(Na_2SO_4)	46 + 32 + 64 = 142	328,100	328,500
($CaSO_4$)	40 + 32 + 64 = 136	317,400	321,800
($MgSO_4$)	24 + 32 + 64 = 120	300,900	321,100
($Al_2S_3O_{12}$)	54 + 96 + 192 = 342	.....	879,700
($(NH_4)_2SO_4$)	28 + 8 + 32 + 64 = 132	283,500	281,100
($MnSO_4$)	55 + 32 + 64 = 151	249,400	263,200
($ZnSO_4$)	65 + 32 + 64 = 161	229,600	248,000
($FeSO_4$)	56 + 32 + 64 = 152	.....	234,900
($CoSO_4$)	59 + 32 + 64 = 155	.....	228,900
($NiSO_4$)	58.5 + 32 + 64 = 154.5	.....	228,700
($Fe_2S_3O_{12}$)	112 + 96 + 192 = 400	.....	650,500
(Tl_2SO_4)	408 + 32 + 64 = 504	221,800	213,500
($CdSO_4$)	112 + 32 + 64 = 208	219,900	231,600
($PbSO_4$)	207 + 32 + 64 = 303	215,700	.....
(H_2SO_4)	2 + 32 + 64 = 98	192,200	210,200
($CuSO_4$)	63.6 + 32 + 64 = 159.6	181,700	197,500
($HgSO_4$)	400 + 32 + 64 = 496	175,000	.....
(Ag_2SO_4)	216 + 32 + 64 = 312	167,100	162,600
($HgSO_4$)	200 + 32 + 64 = 296	165,100	.....

Heat of Formation of Bi-Sulphates.

($KHSO_4$)	39 + 1 + 32 + 64 = 136	276,100	272,900
($NaHSO_4$)	23 + 1 + 32 + 64 = 120	269,100	268,300
(NH_4HSO_4)	14 + 5 + 32 + 64 = 115	244,600	245,100
($HHSO_4$)	1 + 1 + 32 + 64 = 98	192,200	210,200

Heat of Formation of Borates.

($Na_2B_4O_7$)	46 + 44 + 112 = 202	748,100	758,300

Heat of Formation of Cyanides.

Formula.	Molecular Weights.	Heat of Formation.	In dilute Solution.
($Ca(CN)_2$)	40 + 24 + 28 = 92	.....	41,650
(K, C, N)	39 + 12 + 14 = 65	33,450	30,250
(Na, C, N)	23 + 12 + 14 = 49	25,950	25,450
($KAg(CN)_2$)	39 + 108 + 24 + 28 = 199	13,700	5,350
($Fe_7(CN)_{18}$)	392 + 216 + 252 = 860	—256,700	.....
($Zn(CN)_2$)	65 + 24 + 28 = 117	—24,550	.....
($Cd(CN)_2$)	112 + 24 + 28 = 164	—31,850	.....
(Cu, C, N)	63.6 + 12 + 14 = 89.6	—20,375	.....
($Pd(CN)_2$)	106 + 24 + 28 = 158	—49,250	.....
(H, C, N)	1 + 12 + 14 = 27	—27,150 gas	—21,050
($Hg(CN)_2$)	200 + 24 + 28 = 252	—59,150	.....

Heat of Formation of Cyanates.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Solution.
(K, C, N, O)	39 + 12 + 14 + 16 = 81	105,850	100,650
(Na, C, N, O)	23 + 12 + 14 + 16 = 65	105,050	100,250
(Ag, C, N, O)	108 + 12 + 14 + 16 = 150	26,450	.....

Heat of Formation of Ferrocyanides.

($K_4Fe(CN)_6$)	156 + 56 + 72 + 84 = 368	157,300	145,300
($H_4Fe(CN)_6$)	4 + 56 + 72 + 84 = 216	—102,000	— 101,500
($K_2Fe(CN)_6$)	117 + 56 + 72 + 84 = 329	129,600	100,800
($H_3Fe(CN)_6$)	3 + 56 + 72 + 84 = 215	.....	— 127,400

Heat of Formation of Amalgams.

Formula.	Molecular Weights.	Molecular Heat of Formation.	In dilute Mercury Solution.
($Hg_{12}K$)	2,400 + 39 = 2,439	34,600	25,600
(Hg_4K)	800 + 39 = 839	29,700	25,600
(Hg_6Na)	1,200 + 23 = 1,223	21,900	19,000
(Hg_xAu)	x + 197 = 197 + x	....	2,580
(Hg_xAg)	x + 108 = 108 + x	....	2,470

Heat of Formation of Alloys.

Formula.	Molecular Weights.	Molecular Heat of Formation.
($CuZn_2$)	63.6 + 130 = 193.6	10,143
(Cu, Zn)	63.6 + 65 = 128.6	5,783
(Cu_2Al)	190.8 + 27 = 217.8	26,910
(Cu_2Al)	127.2 + 27 = 154.2	21,278
(Cu_2Al_2)	190.8 + 54 = 244.8	17,395
(Cu, Al)	63.6 + 27 = 90.6	1,887
(Cu_2Al_3)	127.2 + 81 = 208.2	10,196
($CuAl_2$)	63.6 + 54 = 117.6	—6,738

[On account of the amount of space already taken up, the discussion of the manner of using these data in metallurgical calculations must be deferred until next month.]

Errata.

In last number, page 104, insert atomic weight of Cadmium, Cd, 112.

On page 106, problem 1, an arithemetical mistake in requirement (2) gives moisture as 0.06, instead of 0.006, and SO^2 as 0.15, instead of 0.015. On page 107, problem 3, under oxygen needed; that for Fe should be 142.8 kilos.

The arithemtical corrections can be easily entered, if anyone desires to correct them. The principles of the solutions in both cases are correct.

Cast-Iron Car Wheels.—A very interesting and fully illustrated article on the manufacture of cast-iron car wheels at the new plant of the Pennsylvania Railroad at South Altoona, Pa., was given in *Iron Age* of January 5. This is the largest and most modern plant of its kind in existence, and is designed to have a capacity of 900 wheels per day, and is a splendid example of the extended use of electric power in modern iron and steel plants. There is an uninterrupted progress of the metal from the cupolas to the annealing pits, each movement being noticeably timed to a nicety, due to the perfect system of handling facilities. The overhead system of application of electric power comprises numerous electric traveling hoists and a special type of electric traveling crane. The tramway system is used chiefly to handle the ladles between the electric hoists and the cupolas, and again to take the hot wheels from the electric hoists to the annealing compartment. Thus, the railways carry the molten metal to the molding floors and carry the completed wheel away, while the electric hoists perform their function during the actual work of casting and molding. At this plant the ever increasing demand of the Pennsylvania Railroad Co. for wheels is to be met by the company itself.

Electric Smelting and Blast Furnace Gases.

By Auguste J. Rossi.

We have had occasion in these columns to treat briefly of the possibilities of electric smelting, as compared to the ordinary metallurgical methods, particularly in reference to the electrical manufacture of steel by what is called the "pig and ore," or the "mixed process:" The direct reduction of oxide of iron for pig iron and steel, electrically, is a subject which has given rise to the most extreme opinions, because we believe it has not always been considered in its proper light.

The economy of such a process does not, indeed, solely depend on the greater or less perfection of the apparatus, though obviously this is as factor of which the importance cannot be ignored, but also on other conditions, which make that, however well combined in all its details of construction the electric furnace may be; however perfect in securing the maximum possible yield from a given charge; however cheaply even the electrical energy may be secured, there exist other objections which limit the application of electricity for such purpose.

It has been said that, in order to obtain the reduction of oxide of iron, there is required a certain amount of energy or its equivalent in heat calories, and that, in the present state of our modern blast furnace development the fuel consumed per ton of iron represents in ordinary conditions of trade, an outlay much less than that at which the electric hp., which corresponds to this amount of heat, could be secured in the most favorable conditions of localities. One metric ton of iron per metric ton of coke is now a current average practice in blast furnaces, and even better results, such as 1850 pounds, and even less, have been obtained per gross ton of metal.

Calculations which can be readily made, and which have appeared at different times in these columns in one form or the other, show indeed that, for an iron ore containing some 57 per cent metallic iron, such ore as is daily smelted in the Pittsburg district, the diverse items which constitute the amount of heat necessary to obtain 1 ton (gross ton) of pig iron (such as reduction of the oxides, melting of slag and product, etc.), counting 1 hp. = 2545 thermal units, or 650 calories (figures adopted by some), there is required at least 140 to 145 hp. If we assume an efficiency of 80 per cent to 75 per cent, this represents some 200 hp per ton, with a margin of 10 per cent, more or less, according to conditions of ores and furnaces. This is a figure which we have stated as the result of several weeks' run in conditions not particularly favorable; the materials used, the metal obtained, the current supplied, being all *measured* and weighed; it is in fact a practical figure which includes all losses.

Ten dollars a year per electric hp. may be considered as an acceptable figure obtainable now; this represents per day an expense of 2¾ cents, and for 200 hp. $5.50, while the price of coke at furnace per ton of iron may be taken at $2.50 to $2.75 in ordinary conditions (it is $2.75 to $3.00 now). This is undeniable, but this assumes that the price of ores, fluxes, fuel, labor maintenance, general expenses, of all these elements of cost, in fact, are the same in both cases; that the electric furnace is to be established side by side with, or in proximity to, the blast furnace. But conditions can be conceived and realized in which, for instance, in certain localities, the ores and fluxes could be had at almost the cost of mining without charges for transportation, so to speak, when the electric horse-power itself could be secured or made available at its lowest figure, all elements which would go to offset the extra cost of the heat-supplying medium. We should mention, also, that the carbon required in an electric furnace in the shape of fuel, does represent but about one-third the quantity required in a blast furnace. In short, calculations made on a given ore, in given conditions of localities, could show in favorable circumstances a parity and even an economy in favor of the electric smelting.

But let this be so; let this be demonstrated, as it is possible to do it in individual cases, the amount of electrical energy required per ton of metal excludes all possibilities other than those of a local limited production.

An output of 500 tons daily, a figure exceeded even by some furnaces in the Pittsburg district, corresponds, at 200 hp. per gross ton day, to 100,000 hp., and even 75,000 hp., assuming the theoretical figure, and were it cheaper to establish such an electric plant than a blast furnace, the possibilities of creating it cannot be thought of in a district but exceptionally and certainly could not be duplicated but rarely, if at all, in the same locality. On this score the attempt to substitute on a large scale, indiscriminately and everywhere an electric smelting plant for our modern blast furnace, is seeking Utopia.

The modern blast furnace is an ideal metallurgical instrument in which the utilization of heat energy approaches perfection in this sense that the metallic product obtained represents, we may say, theoretically, all that a given amount of smelted materials can yield, and no electric furnace could do better on this score. The heat calories of the fuel are accounted for both in the pig iron obtained, slag melted and other contingencies of the process, and in the heat contained in the combustible gases which escape at the mouth. These gases, for many years, have been utilized to heat the blast in special stoves, and also, under boilers, to generate steam, to operate the machinery of the plant, blowing engine and auxiliaries, hoists, pumps, crushers, etc.

It is estimated that, in ordinary conditions, with a good ordinary ore, about one-half of the heat capacity of the fuel is found in these gases. True as this is, it should be remarked, however, that the manner in which these gases are, or have been utilized so far (with only a few exceptions, at least in this country), by far the largest amount of the accountable energy contained in them is now wasted. It is on the utilization of this waste that we desire, in this article, to call the attention, as it opens possibilities for electric smelting in economical conditions which cannot be overestimated, and the importance of this subject is such for the advance of electrometallurgy in general, that we will recall with some details the work done in this direction.

With this part of the gases which is supplied to the hot-blast ovens, we have, of course, nothing to do, and we have only to consider the balance, which is now burnt under boilers to generate steam. To give an idea of the wasted energy resulting from such practice it is enough to remark that a steam motor, working in the best condition, with superheated steam, rarely utilizes more than 5 per cent of the number of calories supplied to the boilers. As a consequence, it not unfrequently happens in some blast furnaces that coke or coal has to be burnt on grates under the boilers to supply the deficiency of the heat supplied by the gases in these conditions. A modern gas motor, on the contrary, as now built, gives readily an efficiency of 25 per cent. Hence, were these blast furnace gases utilized in motors of this kind, there would obviously be a large surplus of power available for other purposes than the requirements of the blast furnace plant. Not only is this efficiency of 25 per cent obtainable now, but the development of the gas engines has rendered possible the construction of machines of 2000-hp., 3000-hp., and even more, and their builders are willing to guarantee: one effective horse-power-hour for every 10,000 thermal units supplied to the motor, which is, indeed, 25 per cent of the theoretical possibilities: since 1 hp.-hour corresponds to about 2545 thermal unit hours, an efficiency of one-quarter, supposes 10,000 thermal units per horse-power hour, or thereabout.

This question of the utilization of the blast furnace gases in gas engines is not new; it has formed the subject of careful examination and actual tests by competent metallurgists in Europe—in England, Belgium and Germany principally—and already the number of engines thus operated with blast furnace gases in Europe represents a total of 200,000 hp. In this

country, for reasons not very clearly explained, it only begins now to receive serious attention and practical application.

Mr. F. duP. Thomson, in a very able and extensive article[1], which has appeared in these columns, has already treated this subject with details both theoretically and practically, and we do not intend in the following to take exceptions to his conclusions, for, though very conservative as they are, they are sufficiently promising to still satisfy the most sceptical. But, owing to the importance of this subject, as bearing on electric smelting and as regards the amount of the available surplus of power, and other data bearing on the same questions, it may be interesting to give the results of experiments and practical tesis made in Europe, as they will be found to agree very well with calculations and data, based on present regular blast furnace practice, or from personal observations, furnished by Mr. Edward A. Uehling[2], the well-known metallurgist, in a memoir bearing date, June, 1903. The general conclusion is the same as that to which Mr. Thomson has arrived, but it presents the question on a still more favorable aspect. For the sake of comparison, we will tabulate at the end of this article the figures given by Mr. F. duP. Thomson, Mr. Uehling, and by European metallurgists.

Whichever figure may be adopted, the most conservative, or a more favorable one, when one realizes what a stupendous amount of power, now wasted, could be created, even with the introduction of the gas engine in only a part of the blast furnace plants, one may well wonder why this application has been so far limited; as leaving aside, of course, the cost of first establishment, all the objections raised against the use of these gases have been successfully answered. The power thus created, transformed in electrical energy, would render possible, beside its utilization for lighting purposes, simultaneously with the manufacture of pig iron in the blast furnace, the establishment of a steel plant, open-hearth or other, for special steels, high-grade steels, etc., in conditions in which the electric smelting could assume a superiority as to cheapness of establishment and quality of product. We may add here, as another application, the manufacture of ferro metals, or metals for seasoning steel.

(*To be concluded in our next issue.*)

Notes on Electrochemistry and Electrometallurgy in Great Britain.

(*From our Special Correspondent.*)

Metallurgical Papers of the Month.

During February the Institution of Mechanical Engineers had no papers of metallurgical interest, so that those in London concerned with metallurgy were able to concentrate their attention upon two papers read before the Institution of Mining and Metallurgy at their meeting on the 16th. Of these, that by Mr. H. F. Marriott on *"Deep Borehole Surveying"* was of considerable technical interest in its succinct description of the electrical devices used for this work. The instruments employed fall into two categories: those which are continuously working, and those which are intermittently working. The general arrangement for either class of work is as follows:

An iron headgear stands over the mouth of the borehole to be surveyed, and the geared drum carrying the electric cable stands at a convenient distance of the rear of this. The drum can be actuated by either hand-power or mechanical methods. In this case a little 5-hp. steam engine is employed. In the headgear is a measuring wheel of extreme accuracy, a vital point where there are known obstructions at various points in the hole. The cable is ½ inch in diameter and contains two highly insulated conductors. It is designed to support its own weight and that of the instrument in a waterless hole of 5000 feet depth, if necessary, though it is unlikely that it would be required to act at this depth. One end of the cable is attached to the head above described, and the other is connected in the interior of the drum to two concentric contact rings, which are placed in communication by means of carbon brushes, with the galvanometer in the continuously recording instrument, or the source of electrical energy in the intermittently recording instrument. The galvanometer itself is of special design, and is calibrated to read, at a pressure of about 4 volts, the make-and-break records of the continuous clinometer, and it is also designed to record the rise in temperature indicated by an electrical thermometer. A small dynamo run by the steam plant is used to supply the necessary current for melting the paraffin wax in the compass and clinometer instrument. The compass and clinometer readings are made by the aid of a protractor of simple design, in which the instrument can be clamped in reverse positions, thus nullifying any errors due to lack of adjustment.

The other paper by Mr. A. Jarman and Mr. E. Le Gay Breretone was mainly intended for analytical chemists. It was entitled *"Laboratory Experiments on the Use of Ammonia and Its Compounds in Cyaniding Cupriferous Ores and Tailings."* Before giving the authors' general conclusions. mention may be made of a series of four tests showing the effect of the presence of ammonium salts in the solutions used for removing copper (Hirsching's process). The percentage of copper extracted varied from 89.8 to 92.2 per cent, an electrolytic assay giving 1.596 per cent Cu, as against an extraction 1.434 to 1.471 per cent Cu. Higher extractions are made when part of the ammonia is present as a salt than when all the ammonia is present as a hydrate. Reviewing Hirsching's method, the authors draw the following conclusions from their experiments:

1. That exceedingly large quantities of ammonia were needed to effect a sufficiently good extraction of copper to allow of subsequent cyanide treatment.

2. That it would be difficult to prevent loss of ammonia by leakage and volatilization.

3. That the recovery of copper and regeneration of ammonia from the solutions would also be costly.

4. That the consumption of cyanide would still be very high after this preliminary extraction of copper.

The authors then proceed to investigate Hunt's method, in which the ore or sands are treated direct by a solution of potassium cyanide to which ammonium hydrate has been added. The gold is thus extracted, together with some copper, and the metals are recovered by electrolytic precipitation, the gold, silver and copper falling to the bottom of the vats as sludge. After describing a variety of tests the authors express their couclusions as regards the Hunt method in the following terms:

1. That for quartzose ores containing cyanicides, particularly copper carbonate, ammonium cyanide is a more efficient solvent for gold than potassium cyanide, in spite of its more rapid deterioration.

2. That ammonium cyanide and potassium cyanide, when suitably protected by ammonia, are about equal in efficiency in the presence of copper carbonate, and are better than unprotected ammonium cyanide.

3. That the quantity of ammonia necessary to suitably protect cyanide solutions from dissolving large quantities of copper depends upon the percentage of potassium cyanide used, and increases with the potassium cyanide.

4. That whilst both ammonia and potassium cyanide are solvents for copper carbonate, yet a combination of these solutions can be obtained which dissolves less copper than does potassium cyanide alone, and it is very near to the point where ammonia exerts its greatest influence in this direction that the maximum gold extraction is obtained (*i. e.*, when using small percentages of ammonia).

[1] Electrochemical and Metallurgical Industry, March, 1905.
[2] Edward A. Uehling, Memoir, Stevens Indicator, June, 1903.

5. That during treatment the copper in solution rapidly reaches a maximum, but afterwards diminished steadily.

The Faraday Society.

The annual meetings announced for February on page 66 of Electrochemical and Metallurgical Industry was unavoidably postponed until March 6. After the formal business, Mr. F. W. Harbord, of the Royal Indian Engineering College at Coopers Hill, will read a paper entitled: "Recent Developments in Electric Smelting in Connection with Iron and Steel." Such a paper must partake very largely of a resumé of the work done by, and report of, the recent Canadian Commission on "Thermoelectric Processes for Smelting Iron Ores and the Manufacture of Steel." The interest will center largely in the discussion with which I hope to deal in detail in my next letter. For Great Britain, with abundant coal, thermoelectric process are either of academic or financial interest—in their practical application they do not appeal to the British iron founder. In greater Britain—particularly in British India—the working of iron and steel by electrothermic processes is one of the problems of the next year or two. Probably Mr. Harbord will be catechized on this point, as his knowledge of Indian conditions is exceptional.

Market Quotations.

The prices ruling for chemicals do not show any great change. Bleaching powder (35 per cent) being quoted at £4.10.0 per ton in Manchester, caustic soda varying from £8.12.6 for 60 per cent to £10.10.0 for 77 per cent. Shellac continued to fall until the middle of the month, when the price fell to 140s. per cwt., from which price it has not further receded. Copper sulphate is now quoted at £22.15.0, a rise of 10s. during the month.

Para Rubber varies very greatly according to quality, nominally the price for fine is about 5s. 3d. Actually, however, some fairly large consignments from the Straits Settlements have been sold in London at 6s. 3d.

Platinum in small quantities has been sold lately at £4.3.6 per ounce. Cleveland pig iron has risen slightly to £2.8.11. Copper has risen slightly to £68.7.6. Tin has been irregular in price, closing at £131.5.0. Ingot and sheet lead have fallen a further 7/6 per ton to £12.10.0 and £13.15.0. Quicksilver has also fallen from £7.15.0 to £7.12.6 per 75-pound flask.

London, March 4.

Society of Chemical Industry.

One of the most interesting meetings held in this season by the New York Section of the Society of Chemical Industry was that of March 24, which was devoted to a discussion of gas as a source of power. The first part of the programme was devoted to the generation of gas, three papers dealing with this subject. Mr. W. F. Case discussed the composition and caloric value of various gases, dealing successively with producer gas (especially the Loomis-Pettibone gas producer, which he considered the only suitable one for bituminous coal), Mond gas and blast furnace gases. Mr. C. G. Atwater spoke on coke-oven gas, his description of various coke-oven gas works being illustrated by a long series of lantern slides. Dr. J. D. Pennock, of the Solvay Process Co., gave a most interesting account of the use of Mond gas in the soda ash works in Detroit, and of the Taylor producer in Syracuse, the results obtained in many years working being entirely satisfactory.

On account of the late hour, the discussion on the utilization of gas for power purposes—which was the second part of the programme—was somewhat brief and in no way as exhaustive as the discussion of the generation of power. Dr. Franz Meyer made a brief remark on the use of illuminating gas for power purposes. Dr. F. Schniewind spoke briefly on the best system for distribution of gas and pointed out that gas would be distributed not only for power purposes, but also for heating and illumination, and that the illuminating effect of the gas depends distinctly on its composition, there being no direct proportionality between the illuminating effect and the calorific effect. Bituminous coal should not be burned as done at present, but first the volatile constituents and by-products should be recovered. Mr. E. A. Uehling spoke on the blast furnace as the biggest power producer. He pointed out the rapid developments in Europe in recent years of utilizing the blast furnace gases for gas-engine driving. He thought the Lackawanna Steel Company, in making the first installation of this kind in this country, had commenced on too big a scale; since it was the pioneer plant it would perhaps have been better if they had commenced with a smaller plant and had used direct current, for the reason that paralleling of alternators driven by gas engines is a serious complication. However, the Lackawanna Steel Company has now overcome all difficulties, and the plant is now working in a perfectly satisfactory manner. The amount of power wasted in not utilizing the blast furnace gases in this country is somewhat like one million horse-power.

A Plain Talk on Trade Journals.

This was the title of a very suggestive lecture, held recently before the American Trade Press Association, by Mr. Arthur Warren, manager of publicity for the Allis-Chalmers Co. Mr. Warren has himself been a distinguished newspaper editor, correspondent and contributor, and has been the first to organize a department of publicity in the industrial world. His address was an imposing array of brilliant and incisive epigrams, somewhat of the Bernard Shaw or James Swinburne style, yet of a distinctly personal flavor. "Who reads the trade papers?" "Some years of experience have forced upon my attention a certain peculiarity on the part of the busy men who haven't the time to read the trade papers. It is this: they know, with an amazing quickness, whenever an article or a paragraph appears, actually or relatively, derogatory to their interests. Hence, it is clear that somebody reads the trade papers." * * * "Advertising space is not the sole instrument used by the Manager of Publicity in the industrial world. It is only one of his tools of trade. Industrial publicity is bounded by limitations which more popular forms of bold advertisement know nothing of. It is one thing to bespatter the universe with the praises of a soap, a breakfast food, a corset, or a pill; to make yourself Governor of a State by first familiarizing a nation with your manly countenance as the guarantee of the perfection of a boot; and it is another thing, a very different thing, to increase, by advertising, the sales of reciprocating engines, steam turbines, hydraulic turbines, gas engines, mining machinery, flour mill machinery, rock and ore-breakers, saw-mill machinery, hoisting and pumping machinery, electric generators, motors, transformers, controllers, and the like. The advertiser of the pill, the boot and the soap, sings his song to the world as it runs and travels and plays, and he sings it with a sledg-hammer accompaniment, a sort of anvil chorus. He invents a phrase and repeats it until you know it by heart, and speak it in your dreams; he entertains you with pictures, and amuses you with puzzles. Turn where you will you cannot escape him. He is omnipresent. If you believe the experts who expound their art so solemnly, he is omniscient. It doesn't matter what he says, it's all in the way he says it. The industrial advertiser has not this advantage. He speaks to a sophisticated audience, an audience trained by daily experience with the sharp cares of business, and founded, in many thousands of cases, upon a technical education along the very lines upon which he seeks to address it. The industrial advertiser speaks to a public which, to say the least, is quite as likely to know as much about the wares he advocates as he knows himself. And there is no one so cautious as your tech-

nical man when he touches publicity. I am not altogether sure that technical men wholly approve of publicity, although they may revel and wallow in the fruits of it. Submit to one of them the proof of an engine or electrical ad. His view-point is not yours. He will cover all your white space with black dissertations on shunt windings and efficiency curves, with talk of tests and measurements, weights, pressure, and electromotive force, while you are trying to tell in a few terse phrases what the machine is, what it is for, and what it will do." * * "The Publicity Man's business is publicity as applied to the business of selling machinery, not as applied to the art of constructing it. And the trade and technical press is a vehicle for industrial publicity. And because of that it flourishes" * * "Industrial publicity is a new thing. It is still young and learning to walk. I sometimes think that no one appreciates the possibilities of publicity so much as the Publicity Man himself. And its appreciation in the industrial field is a thing of yesterday morning. The day before, no one thought of it. But 'the old order changeth, yielding place to new.' For all that there is no industry, among all those represented on this occasion, that has all the publicity it needs; there is none that has begun to more than taste the fruits of advertising; there is no industry, there is no metal-working manufactory that spends money enough on publicity. Manufacturers are, no doubt, accustomed to say 'Publicity cost so and so.' They are not yet accustomed to think 'Publicity earns so much.' That is a part of the lesson they have yet to learn. And they will learn it, and appreciate it. Yet I once knew a manufacturer who said: 'We are getting too much publicity.' There are evidences that his opinion is changing." * * * "Every advertiser should buy and handle his own advertising space, design his advertisements, and, in short, carry his own Department of Publicity." A long part of Mr. Warren's address was devoted to a discussion of circulation. "In circulation we recognize two elements that make value, quality and quantity; and these two are the product of a third, character—the character of the publication."

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

By George P. Scholl, Ph. D.

Electric Furnaces and Furnace Products.

Method of Agglomerating Magnetic Ore.—E. Gates, Chery Chase, Md. Patent 780,716, January 24, 1905. Application filed January 14, 1901; renewed November 5, 1902.

The invention relates to the treatment of magnetic iron ore sands, in order to bring about their agglomeration into larger lumps, and thus make them available for use in a blast furnace. The operation is performed by allowing the particles to be fused by an electric arc sprung between two inclined feed troughs, constructed of conducting material. The opposing ends of the inclined troughs are brought together when the process is started, and are then drawn apart, so as to establish the arc between them, and the magnetic sand is fed from hoppers upon the inclined troughs, down which it slowly descends. The falling streams of ore approach each other and the arc passes between them along the line of least resistance. Care is said to be taken that the sand is fed at a speed in excess of that necessary for the utilization of all the energy of the current. As a result, the sands are agglomerated to small lumps, varying from the size of a grain of wheat to that of a bean. They accumulate in a tray located beneath the arc, where they are partially cooled, but from which they overflow and drop into a hopper, while still hot. The fused particles are thus brought into contact with the partly cooled materials in the tray and form lumps of magnetite of increased size. The treated material then passes from the hopper into a revolving screen, where the lumps are screened out and from which the untreated sand is returned to the apparatus for retreatment.

Electric Furnace.—J. M. Morehead, Chicago, assignor to Union Carbide Co. Patent 782,917, February 21, 1905. Application filed October 19, 1903.

The furnace is specially intended for the production of calcium carbide, and is an improvement on the inventor's former furnace, which comprised a hood enclosing the electrodes, the charge being fed around the hood in order to exclude the air and to direct the waste gases into the hood. The inventor states that he has found that the charge may advantageously be fed directly into the hood, and that the waste gases will still collect in the hood, although obliged to pass through a considerable body of the charge before they reach the gas outlet. This upward movement of the gases is believed to be due to the fact that the gases rising from the zone of reaction at the lower end of the electrodes tend to work upwards through the charge along the electrodes, and that they will therefore pass along the electrodes through a path of greater length than through the charge itself. It is stated that this new construction enables the furnace to be operated with great regularity and without filling the surrounding atmosphere with dust from the finely divided charge. The furnace, shown in vertical section in Fig. 1, is of the Horry type, the working chamber consisting of a wheel 1 with removable cover plates 2. An iron hood, carrying electrodes 4 fixed in vertical adjustable holders 5, depends into the open end of the working chamber. The top of the hood is closed by a cast-iron plate, containing a chamber for water-cooling purposes, with water inlet and outlet 8 and 9. Parallel openings 10 are provided for receiving the electrode-holders, and a refractory luting 11 is filled into the space left between the walls of the openings and the holders, after the electrodes have been adjusted to their proper height. A pipe is provided for the purpose of introducing the charge, behind the electrodes, and another pipe serves for withdrawing the gases, and is located in front of the electrodes. During the operation the electrodes are adjusted to their proper height and the charge is fed in until it fills a considerable portion of the hood and thus seals its lower end.

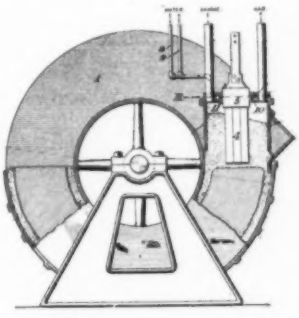


FIG. I.—MOREHEAD'S CARBIDE FURNACE.

Process of Smelting Refractory Ores.—E. F. Price, Niagara Falls, assignor to Union Carbide Co. Patent 782,922, February 21, 1905. Application filed October 19, 1904.

The process relates particularly to the production of calcium carbide, and reference is made to patent 750,096, January 19, 1904, (described in Electrochemical Industry, Vol. II, page 109), obtained by Mr. Cowles. In the latter the current is passed through a charge in a furnace with downwardly converging side walls, in order to continuously increase the current density as the charge reaches the lower parts of the furnace. The electric current is passed by means of carbon electrodes, the upper electrode being constituted by a carbon ring

in the upper part of the furnace, while the carbon hearth of the furnace forms the other electrode. It is claimed by the inventor of the present patent that carbon electrodes of the size and character required for this operation are expensive and perishable; that it is difficult to obtain large blocks of uniform composition and that the carbon is rapidly worn away by the attrition of the charge and its products, and in many cases reacts chemically with them. The inventor therefore proposes to interpose an electrically conductive charge of a calcium compound and a reducing agent, specifically a mixture of lime and carbon, between water-cooled metallic electrodes, and to smelt it by the heat generated by the resistance of the charge. The lower electrode is preferably constituted by a body of molten iron or other metal which will not combine with the carbide in the lower part of the furnace chamber. The molten iron in that case is stated to purify the carbide by withdrawing the silicon from it, which latter is one of the normal impurities in carbide, by which means the iron is converted into ferro-silicon. The furnace is shown in vertical cross-section in Fig. 2. It comprises a vertical stack with a body 1 of refractory material, such as magnesia or siloxicon fire brick, surrounded by a water-jacket and supported upon a cast-steel plate, with cooling chamber and electric terminal. The upper electrode is supported upon this body and insulated from it by a layer of non-conducting refractory material. It is constituted by an iron ring, surmounted by a water-jacket, the inner surface of the ring being bare, in order to insure contact with the charge. Bell and hopper and outlet flue are provided as usual, in an iron ring, which covers the top of the furnace. When the operation is started, the furnace is filled with a mixture of lime and carbon until its upper portion lies in contact with the electrode ring, initial current paths being provided if the charge is normally a poor conductor. The conductivity of the charge may be increased by using a mixture containing large pieces of coke, which lie in contact with each other at various points, and thus furnish direct paths for the current. A current of sufficient amperage is then passed, the temperature in the charge rising gradually downwardly on account of the decreasing cross-section of the furnace, down to a zone where the materials react to form carbide and the latter is brought into a molten condition. In normal operation the major portion of the body of iron will be maintained in a molten condition by the heat of the charge, the lower portion being pasty or solid, owing to the influence of the cooling water. The molten carbide accumulating upon the iron is intermittently or continuously tapped off through the tapping-hole, and fresh material is fed in as required. The waste gases preheat the charge and after passing out through a pipe, are used as fuel. The silicon withdrawn from the molten carbide by the iron converts the latter into ferro-silicon, which is removed from time to time by a tap-hole, more iron being introduced with the charge.

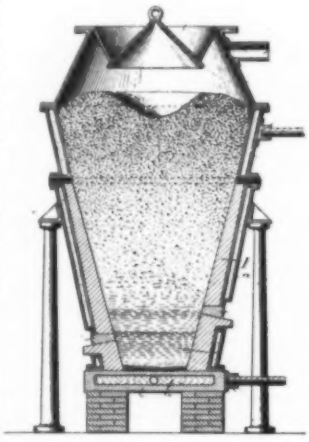


FIG. 2.—PRICE CARBIDE FURNACE.

Electric Furnace.—G. O. Seward, Holcomb's Rock, Va., assignor, to Willson Aluminum Co. Patent 783,736, February 28, 1905. Application filed May 3, 1904.

The furnace illustrated in this specification is of circular cross-section, with two electrodes of opposite polarity passing through the top cover. The improvement claimed in the invention consists in the construction of this top cover, in such a manner that electrodes of different sizes can be used with the same cover; that repairs are facilitated and the damages to the top due to the establishment of arcs from the electrodes are lessened. The cover used by the inventor is flat and made hollow for the purpose of circulating water for cooling. Apertures are provided in the cover for the electrodes, which apertures are somewhat larger than the electrode. Each electrode is fitted with a horizontal flange overlying the top of the cover and a packing ring of asbestos, asbestos cloth or other fireproof insulating materials is introduced between the lid and the cover. The electrodes are thereby insulated from each other, and the construction is stated to have the further advantage that in case of an accident, and the establishment of an arc from the electrode, it is only the special lid of that electrode which is damaged, and a new lid can be supplied very quickly without even removing the furnace cover.

Process of Utilizing Scale from Calcium Carbide Ingots.—T. R. Edmands and J. Scales, Sault Ste. Marie, assignors to Union Carbide Co. Patent 784,255, March 7, 1905. Application filed December 22, 1904.

The inventors propose to utilize the carbide scale by treating it with sufficient water in order to decompose the carbide it contains. The residue is then dried and smelted in the usual carbide furnace after sufficient carbon has preferably been added to make the total carbon contents of the mixture equivalent to its contents of lime. It is stated that in practice it is desirable to crush the scale into small pieces, and to effect the decomposition of the carbide contained in it in an acetylene generator, provided with agitators, so as to insure contact between the particles of the scale and the water. The gas is collected and utilized and the water is driven off from the residue by means of any suitable drier. The residue, when mixed with sufficient carbon, is then either smelted directly or added to the normal furnace charge of lime and carbon.

Electrolytic Production of Metals and Compounds.

Electrolytic Process of Reducing Metallic Sulphides.—C. E. Baker and A. W. Burwell, Cleveland. Patent 782,894, February 21, 1905. Application filed July 1, 1904.

The process depends upon placing the metallic sulphide or sulphides in contact with or in proximity to the anode of an electrolytic cell in which a molten chloride is electrolyzed. The chlorine given off there replaces the sulphur in the sulphide, and the chloride thus formed melts and replenishes the electrolyte. The operation is carried out in a vessel shown in cross-section in Fig. 3. It consists of an iron vessel, the sides of which are provided with a refractory lining of magnesia fire brick or other non-conducting material. The anode is supported on shoulders, provided on opposite sides of the lining. The bottom of the vessel constitutes the cathode, and is connected to a source of current. The anode consists of a number of parallel bars 5, preferably of Acheson graphite, which are spaced apart and enter a horizontal bar conductor, from which rises a terminal post, which in turn is connected to the supply conductor. The bar and the post are preferably made of graphite. The vessel has a cover luted to it by means of asbestos cement. A charge opening and gas outlet are provided as usual, and a bushing insulates the post from the cover. In operation the cell is filled with an electrolyte 14 of molten chloride; for example, lead chloride. The metallic sulphide, for instance, lead sulphide, is preferably supported on the anode itself, although a separate shelf in proximity to the anode may be em-

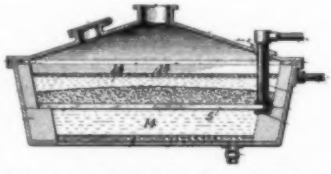


FIG. 3.—APPARATUS FOR REDUCING SULPHIDES.

ployed. The chloride is then electrolyzed, the lead being deposited as a molten layer on the bottom of the apparatus, whence it may be removed by means of a cock. The chlorine set free at the anode is stated to decompose the sulphide, freeing the sulphur and reproducing lead chloride. The sulphur may either be allowed to accumulate as a floating layer 15 on the surface of the electrolyte, and may then be skimmed off, or it may be volatilized and driven off through an opening. By regulating the current density so that an excess of chlorine is generated, sulphur chloride (S_2Cl_2) may also be formed. It is stated that a chloride of a metal other than that of the sulphide to be treated may be used as the electrolyte, and that, for example, molten zinc chloride has been employed for the treatment of lead, as well as zinc sulphide. Mixed or complex sulphides may also be treated.

Apparatus for Cyanide Treatment.—E. L. Oliver, Oakland, Cal. Patent 784,120, March 7, 1905. Application filed June 20, 1904.

The object of this invention has been to overcome the difficulty of scouring the amalgamated plates used for recovering the values from a cyanide solution, when slimes or sands are agitated with cyanide directly in the vessel in which electrolysis is being carried out. The apparatus is illustrated in Fig. 4, in vertical section. It consists of a tank *A*, with a conical bottom, provided with a discharge, which is controlled by a gate *b*. A casting *d* with openings *d'* is bolted to the bottom, and a pipe *D* fits into it. An air pipe leads from an air compressor into the interior of the pipe *D* and terminates a short distance above the bottom of the latter. Electrodes *E* and *F* (the latter not shown, but parallel to *E*) are located in the upper part of the tank, and a large number of them are provided, so as to provide a large surface. The electrodes *E* are the anodes, and are made of iron plates or other suitable material, while the electrodes *F* are the cathodes and made out of copper plates coated with mercury. The electrodes are suitably supported on block and strips *G* and *H*, and boards *T*. By means of a trough the copper plates are automatically provided with mercury, thus lengthening their period of use before they have to be taken out to remove the deposits. The operation of the apparatus is as follows: The material to be treated and the cyanide or other solvent solutions are filled into the tank until they reach above the level of the pipe *D*. Air is then turned into the pipe *C*, which forces the solution out of the pipe *D* and draws it in through the apertures at the bottom, thus maintaining a continuous circulation past the electrodes. At the same time the electric current deposits the gold and silver in the solution on the amalgamated plates. It is stated that this agitation and circulation of the solution can be performed as gently as desired, and that though efficient contact is made with the amalgamated plates, there is no scouring action upon them.

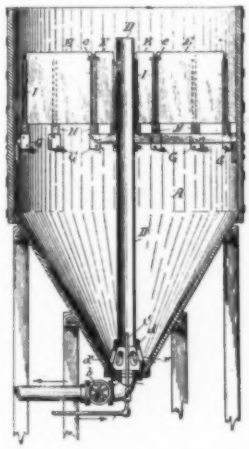


FIG. 4.—CYANIDE APPARATUS.

STORAGE BATTERIES.

Electrolytic Production of Superoxides in Alkaline Solutions.—H. Rodman, Philadelphia, assignor to Electric Storage Battery Co. Patent 782,989, February 21, 1905. Application filed May 17, 1902.

The object of the invention is to accomplish the production of metallic superoxides, such as that of nickel, directly from the metal, for the purpose of using them as active material in storage batteries. In the case of the production of nickel superoxide, the metal is constituted the anode, an iron cathode being used. As under ordinary conditions the nickel would be passive in an alkaline solution, a very dilute alkaline solution is used as an electrolyte, to which may be added an acid radical, either acid or a salt containing the desired radical being available for use. As example of such substances may be mentioned inorganic salts or acids, as sodium perchlorate, sodium nitrate, etc., also organic salts or acids, such as tartaric, acetic and oxalic acid, etc., and sodium salicylate, etc. An electrolyte suitable for the practice of the invention would be a one-half per cent solution of caustic potash, either by itself or containing one-tenth of one per cent of sodium perchlorate. A current of about half an ampere per square inch of anode surface, more or less, is appropriate. The temperature of the solution may be less than 100° F. Instead of using such a very diluted solution as mentioned above as an electrolyte, the bulk of the solution may be of ordinary strength, and only that part of it which is in immediate vicinity of the anode, may be made dilute. This may be accomplished by placing fibrous material, such as blotting paper, adjacent to the anode, which acts like a diaphragm, and permits the portion of the solution nearest to the anode to become dilute by the electrolytic action.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

PRACTICAL APPLICATIONS.

Electrometallurgy of Iron and Steel.—Charles Combes, in a paper before the *Societe d'Encouragement pour l'Industrie Nationale*, discussed P. L. T. Héroult's work in the electrometallurgy of iron and steel. Since Dr. Héroult's work in this field has been dealt with repeatedly and at length in our columns (Vol. I, p. 63, 287, 449, 461, 467; Vol. II, p. 408, 481) we give only those parts of Combes' address which contain new information.

Ferro-Chrome.—For making ferro-chrome, Héroult uses a crucible lined with chrome ore, and the two electrodes are suspended into the slag above the molten bath (in the same way as in his steel furnace). By adjusting properly the amount of carbon for reduction, he makes ferro-chromes which contain 2 to 6 per cent. of carbon. It is possible to reduce the amount of carbon still further below 1 per cent by a refining reaction in the bath; for this purpose an excess of chromite is added to the slag, and the excess of carbon is removed by the reaction between the molten bath and the oxidizing slag. The "soft" ferro-chrome thus produced differs considerably from the "hard" ferro-chrome, containing 9 or 10 per cent. C. The soft alloy is white and compact, is not brittle and presents on fracture a crystalline structure which is the more developed the less the content of carbon. Sometimes parts of the alloy are obtained in form of magnificent crystals up to 1 cm. in size. They do not represent a strictly defined chemical compound. On the contrary, each crystal is made up of a series of concentric envelopes of different nature, one fitting always exactly on the next one. These alternating layers consist of two essentially different compositions; the one being soluble in dilute hydrochloric acid and containing no carbon in notable quantities; the other being insoluble in dilute hydrochloric acid and containing 5.4 per cent of carbon. The relative proportions of chromium and iron are a little different in the soluble and insoluble layers. In 100 parts of

alloy, the layers which dissolve in acid have a composition of 46 per cent. Cr, 54 Fe, 0.03Si, 0.16Mn. The layers, insoluble in dilute acid, are soluble in hot concentrated hydrochloric acid, a residue of carbon being left. The composition is given as (Fe 1.5, Cr 1.5) C.

Reduction of Pig Iron from Ores.—Concerning Héroult's work in this field, Combes deals especially with the "economizer" and with another process for saving electrical energy which were already mentioned in our Vol. II., p. 408. The idea of the economizer is given as follows: "For the reduction of 1 kg. of pig iron, 2000 to 2500 calories are required for the reduction as well as for the fusion of the bath and slag. For the reduction, 330 grams of carbon are required, CO being taken as basis of calculation. The heat developed in the furnace during reduction is $24 \times 0.33 = 792$ calories. The difference amounting to 1200 to 1700 calories must, therefore, be supplied by electrical energy. But the burning of CO can furnish $0.33 \times 5600 = 1800$ calories. Let us now suppose that this CO is burned in a special apparatus in which the ore is fused and even overheated before it comes into the reduction furnace. In this way it is possible to recover the greatest part of the calories which are lost." The second method of saving electrical energy was briefly outlined on page 408 of our Vol. II., apparatus for this purpose being described on p. 467 of our Vol. I.

Steel Making.—As repeatedly described (f. i. Vol. II., p. 481) Héroult removes the sulphur and phosphorus from the molten bath by means of artificial slags, which are renewed as often as necessary (one method, for instance, being based on the action of CaC_2 added to the slag). The metal is afterwards recarburized by means of "carburite." Carburite is "a mixture of iron and carbon filings or shavings with carbon and the required quantity of binding material. This product is dense enough to pass through the layer of slag and come in contact with the metallic bath in which it dissolves rapidly on account of the high temperature of the arc. It is thus possible to realize the deoxidation and recarburation of the metal without addition of any foreign material such as silicon and manganese, and the required composition can be obtained very exactly by weighing in advance the quantity of carbon to be used." The author describes some mechanical details of construction of the furnace, all of which were already described in these columns.

Induction Furnace.—In the discussion of Combes' paper, Messrs. Lodin, Saladin and LeChatelier participated; the latter paying a special tribute to the services rendered by Héroult to the French industries. Saladin spoke on the induction furnace and remarked that not Kjellin, but Ziani de Ferranti (English patent 1885, Jan. 15) is the original inventor of this type of furnace. (See also the letter in the correspondence columns of this issue.) Schneider & Co. (Le Creusot) have devised a modification to produce circulation in the furnace. As described in our Vol. II, page 283, they construct the furnace in form of a tube of small cross section, which communicates at both ends with a chamber of large dimensions containing the greater part of the molten metal. The latest form of the furnace, as used by Schneider & Co., is shown in Fig. 1. It consists of a large reservoir communicating with two straight tubes, in connection with a three-core transformer. The complete paper of Combes, together with the discussion, is given in the January issue of *Revue de Metallurgie*.

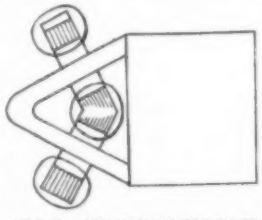


FIG. 1.—INDUCTION FURNACE.

Electric Furnace in the Iron and Steel Industries.—In *Cassier's Magazine* of March, H. Allen gives some calculations with reference to the thermochemistry of iron ore reduction and steel making in the electric furnace. The author concludes that "a study of the thermochemistry of the production of iron and steel by means of the electric current shows that when carried out on a system as nearly as possible in accordance with that required by theory, the capabilities of profitable applications only require scientific treatment to prevent an excessive charge for electric current from turning the scale between profit and loss, even in the field of the blast furnace."

Detinning of Tin Scrap.—In the *Elektrochemische Zeitschrift* of February, H. Mennicke begins to give a review of the industry detinning tin scrap by electrolysis since 1902. In the present instalment the author deals especially with the commercial side of the question. He points out that the financial success of a detinning plant depends no longer on the soundness of the process alone—detinning with an alkaline electrolyte is cheap and easy—but rather on the geographical situation of the works, on the supply of scrap, and especially on a good buyer of scrap. The pioneer German plants have formerly earned considerable dividends by this industry, but since that time the price of scrap has been greatly increased by competition, new works having been erected. The author does not think that the outlook is encouraging for the new works, since the whole supply of scrap is in the hands of the old firms.

Electrolytic Ampere-Hour Meter.—In the *Zeit. f. Elektrochemie*, March 10, H. Danneel discusses mercury voltameters. Mercury is specially suitable on account of its high equivalent weight, and for the reason that from mercurous nitrate solutions it is deposited monovalent, Faraday's law being fulfilled with great accuracy. The mercury which is deposited may be weighed, as in Danneel's arrangement in which the cathode is suspended on a balance, or its volume may be measured, as in the Wright meter. A modification by Lux of the latter is described; Fig. 2 shows the arrangement of the meter from the side and from the front. The anode *A* is on the top, and is always at the same distance from the platinum cathode *B*. The anodic mercury *A* rests on a narrow-mesh wire screen from which it does not drop off on account of capillarity. As soon as some mercury goes into solution from the anode *A* it is replaced by fresh mercury from the reservoir *D*, while some of the solution above *A* enters into *D*. The arrangement of the anode above the cathode provides automatic circulation of the electrolyte, the heavier anode liquid flowing downwards while the lighter cathode liquid rises. The first mercury which is deposited in the platinum cathode *B* adheres on it. When some mercury has accumulated on it, all mercury which is deposited afterwards drops into the funnel *J*.

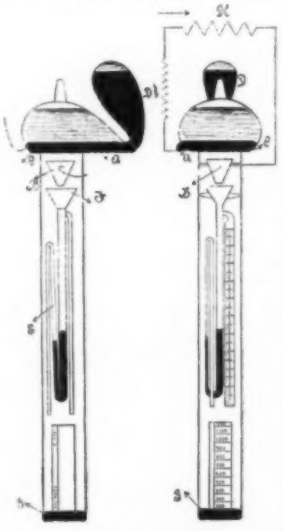


FIG. 2.—METER.

In the first moments the reading of the meter will, therefore, be too low. But after it has been in service for a short while, as much mercury drops into *J* as is deposited on *B*. The mercury accumulates gradually in the first and second vertical tubes until it rises to the top of the second tube. Then by a syphon action the whole mercury flows off through the third tube, the arrangement being quite clear from the illustration. The mercury then accumulates in the lower part of the apparatus. There are two scales, the lower scale acting like the hour hand of a watch, the upper scale like the minute

hand. The meter is connected in parallel with an adjustable resistance *R*, which is inserted in the circuit carrying the current to be measured.

Kryptol Electric Furnaces.—In view of the increasing use of kryptol as resistance material in electric furnaces, we give the following references to publications on this subject, the principle having been repeatedly described in our columns. A. Voelker described various applications of kryptol in *Verhandl. d. Ver. z. Beförd. d. Gewerbfleisses*, 1904, Heft 5, a greater number of apparatus using kryptol being here illustrated. A shorter account which is also illustrated, was given by W. Bermbach in *Elek. Anzeiger*, 1904, Nos. 60, 61.

Theoretical and Experimental.

Bromine.—The potential of a halogen electrode depends on three variables, the concentration of the free halogen, the concentration of the halogen ions, and the temperature. The *Zeit. f. Elektrochemie*, January 27, contains a long paper by F. Boericke on experiments in which he studied the following problems: The solubility of bromine in potassium bromide and hydrobromic acid solutions; the potentials of bromine in potassium bromide and hydrobromic acid solutions; and the anode potentials during the electrolysis of neutral potassium bromide solutions. The main results may be summed up in the statements that the strong solubility of bromine in water, or in bromide solutions permits the reversible discharge of bromine ions at a platinized platinum anode at such anode potentials, which even with a higher current density are still below the bromine potential corresponding to saturation. On the other hand, at a polished platinum anode, on account of the changes of the electrode condition, very much higher voltages than correspond to the bromine potentials are required to pass a strong current through the anode.

Electric Combination of Hydrogen and Oxygen.—Rev. P. J. Kirkby has formerly described some experiments which exhibited the effect of passing electricity with a luminous discharge through hydrogen and oxygen, mixed in equivalent proportions and at low pressures. The experiments showed that a partial union of the gases into water vapor is the result, and that if the pressure and the potential difference of the plates during the discharge remain steady, the quantity of water vapor formed, is proportional to the quantity of electricity which has passed—in other words, to the product of the current and the time. The same investigator has now made further experiments to study; first, how changes in the distance between the parallel electrodes, between which the discharge takes place, would affect the results, and secondly, whether the same effects would be observed if non-oxidizable plates were used. His paper is published in the January issue of the *Philosophical Mag.*, and abstracted in the Lond. *Elec.*, January 27. He prepared the gases simultaneously by the electrolysis of pure barium hydrate contained in a small sealed glass vessel. By this method the gases were generated free from hydrocarbons, ozone and peroxide of hydrogen. The mixed gases were dried and admitted to a glass bell jar containing the parallel plates between which the current was to be passed. The parallel plates were connected to the poles of a battery of small lead cells, ranging up to 1080 volts, through a high resistance and a sensitive ammeter. In this way the quantity of electricity passed through was known, while the pressure before and after its passage were determined by McLeod gauges. The results showed that in spite of great variations in the pressure and in the distance between the electrods, the lowering of pressure in millimeters of mercury bore to the quantity of electricity in coulombs passed through the hydrogen and oxygen a nearly constant ratio, amounting to about 2, and ranging from 1.6 to 2.4. The distances experimented with, ranged from 0.25 cm. to 1.5 cm., and the pressures from 1.1 mm. to 9.3 mm. of mercury. Within these limits, at all events, from six to nine times as many molecules of water vapor are formed as pairs of ions. This result is independent of the capacity of the apparatus and of the nature of the electrodes.

Metallurgy.

Copper and Brass.

L. Guillet, in *Revue de Metallurgie* for February, publishes a long article of 24 quarto pages on the copper alloys and special brasses. The matter was the subject of a lecture given before the Société d' Encouragement pour l'Industrie Nationale on the 13th of January. A diagram is given showing the variation of tensile strength and extensibility for brasses up to 50 per cent zinc. The tensile strength of rolled brass, annealed, reaches its maximum of 57 kilos for square millimeter (81,000 pounds per square inch) at 30 per cent of zinc, the elongation its maximum of 49 per cent at 44 per cent zinc. Very beautiful photomicrographs are given of alloys with 8, 33, 41, 48, 57, 79, 89 and 93 per cent of zinc.

Under the heading of "special brasses," are considered those with lead, tin, aluminium and manganese. Using lead, up to 5 per cent, can be added without liquation, the micrographs showing that the lead enters mechanically between the crystals of brass, splitting large crystals into small ones, and so facilitating rolling and forging. Above 5 per cent the lead tends to liquate out in irregular masses, but the brass can be worked and rolled with up to 7 per cent. It is to be regretted that Mr. Guillet stopped with this amount, and did not investigate these very useful and important brasses with up to 25 or 30 per cent of lead, which make such superior bearing metal.

The study of the "Kalchoids," the tin-zinc-copper alloys did not reveal any new information. The naval brass of England and France contains 60 to 62 parts copper, 37 to 39 parts zinc and 1 to 1.5 parts tin.

In manganese brasses, it was found that up to 10 per cent of manganese it acts simply as so much zinc; an alloy with 45 zinc and 10 manganese, for instance, acting like a brass with 55 per cent zinc.

Aluminium brasses were particularly studied. Using brass with 40 per cent of zinc, the introduction of 0.5 to 1 per cent of aluminium gives a fine golden color; this color persists to about 5 per cent of aluminium, and then becomes rose; at 7 per cent a fine rose color is produced, and the alloy is dichroic; that is, it is gray, seen at a certain angle of incidence; at 10 per cent the alloy becomes as white as silver. Up to 4 per cent it can be worked hot; above that it is very difficult to work; above 6 per cent it can no longer be rolled; the 10 per cent alloy is so brittle that it cannot even be polished. As for the mechanical properties in the testing machine, the addition of aluminium copper-zinc alloy acts like a strong increase of zinc content, each per cent of aluminium acting somewhat like an increase of 3.5 per cent of zinc.

Concerning the quenching of copper-tin bronzes, the maxima of tensile strength and elongation were obtained by quenching from the temperatures, as given:

Copper.	*Tin.*	*Maximum Strength.*	*Maximum Elongation.*
95	5	450° C.	400° C.
91	9	700° C.	800° C.
87	13	725° C.	725° C.
84	16	600° C.	650° C.
79	21	600° C.	650° C.

In every case except the first (5 per cent tin), the strength and elongation decreased for quenching temperatures up to 400-450° C., and then rapidly increase to maxima at the temperatures given.

Gold.

Cyanide Process.—Cyanidation in the United States is discussed by G. H. Fulton in the *Eng'ing and Mining Jour.* of January 5. He first deals with electric precipitation. At the present time there are two such plants in the United States and one in Mexico. This recovery does not compete with zinc precipitation, as applied to ordinary gold-bearing cyanide;

it finds its application to foul solutions carrying much base metal, such as are obtained from weathered tailing, or from silver ore, where the bulk of the precipitate becomes very great. The Malm process is in use by the Gold Cotd. Mining Co., treating the old tailing of the Empire mine, near Maryville, Mont., as described by Mart. W. Anderson. The precipitation takes place in boxes (9 by 10 feet in cross-section, and 5 feet deep), divided longitudinally into three compartments. All electrodes are of No. 18 iron. The sub-plates are coated on the positive side with graphite. The current density is 0.25 amperes per square foot of cathode surface. The solution does not circulate through the boxes, but is permitted to stand in contact for 1.5 to 2 hours, when it is drawn off and replaced by fresh solution. The deposit forms in flakes on the negative side of the sub-plates, adhering to them loosely. It consists on the average of 51 per cent copper, 3.2 per cent gold and silver, some arsenic, antimony, lead and considerable lime. The author then describes E. M. Hamilton's electric precipitation in the Butters' plants at Minas Prietas, Mexico, and at Virginia City, Nev. This process was described at length in our Vol. II, p. 131.

The author then deals with the crushing of ore. He remarks that Chilean mills are finding increased application, both for large and small units, even replacing stamps. This is true, both for mills, which cyanide direct, and also for those which amalgamate previous to cyanidation. The roller mill presents the advantages of lower cost of installation, less power consumption, and large crushing capacity for small space. When they are used, rock breakers and coarse rolls prepare the ore for subsequent treatment. The author then discusses the treatment of slimes. While the ordinary decantation process or some modification of it is still standard in this country, yet its imperfection involving the loss of soluble gold and cyanide is becoming recognized and experimentation points to filter pressing. "Fine grinding, for high extraction and quick solution of of gold and silver, has been emphasized recently; instead of the prevention, the production of slime seems to be favored. In this connection, however, the question arises: What method of slime treatment is available? With most ores, the slime will have a higher value than either the sand or the original ore, and unless the filter press commends itself on the score of cost, excessive sliming presents a doubtful advantage, owing to the loss of dissolved gold inherent in the decantation process; in the Black Hills the loss in dissolved gold ranges from 5 to 9 per cent of the assay value. This fact has cooled any ardor to crush finer, and with ore that is too low grade to permit filter pressing, the production of an excessive amount of slime is to be avoided. It is probable, however, that the filter press will be modified in the near future so as to reduce the cost of operation and make it available for low-grade slime." The author then deals with the precipitation and treatment of precipitate, with the increasing use of sodium cyanide, and gives a review by States.

Electroplated Copper Plates in the Battery.—The *Journal* of the Chem., Metal. and Min'g Soc'y of So. Africa, of October, 1904, contains a paper by F. W. Cindel, advocating the introduction of silver-plated copper plates into metallurgical practice on the Rand. The author gives an outline of the practice of the Homestake Gold Mining Co., in South Dakota, where the ore under treatment is a hornblende, garnetiferous schist or slate infiltrated with free silica and pyrites; in a 200-stamp mill, there are 160 plates with an amalgamated surface of 8640 square feet. He makes a comparison with Rand practice where the area of plates for a 200-stamp mill is only 2400 square feet; he therefore recommends additional plates.

Miscellaneous.

Mineral and Metal Statistics.—The issue of Jan. 5 of the *Eng. and Mining Jour.* contained the carefully prepared and very elaborate annual statistics on mineral and metal production during the past year. The following figures give the United States production in 1904, the figures in parenthese giving the output in 1903 for the sake of comparison.

Non-Metallic.				
Arsenic, white	498	(611)	short tons	$29,504
Barytes	22,502	(50,397)	short tons	74,736
Bauxite	48,966	(48,087)	long tons	189,325
Bromine	897,100	(597,000)	pounds	313,985
Carborundum	7,000,000	(4,760,000)	pounds	700,000
Cement, natural hydraulic	5,000,000	(7,030,271)	bbls. of 300 lbs.	2,500,000
Cement, Portland	22,000,000	(22,342,973)	bbls. of 380 lbs.	20,900,000
Coal, anthracite	69,749,962	(75,288,206)	short tons	149,990,870
Coal, bituminous (including cannel)	273,774,922	(277,076,986)	short tons	316,763,701
Coke	23,500,000	(25,262,360)	short tons	56,400,000
Cobalt, oxide	100,000	(120,000)	pounds	215,000
Copper sulphate (including a small amount made from metallic copper)	63,309,394	(43,124,454)	pounds	2,849,484
Copperas (including only that marketed as copperas)	12,404	(20,240)	short tons	868,315
Crushed steel	395	(378)	short tons	53,300
Fluorspar	26,180	(42,523)	short tons	153,081
Garnet	3,363	(4,413)	short tons	99,463
Graphite, artificial	3,195,251	(2,620,000)	pounds	221,686
Graphite, crystalline	4,594,282	(4,525,700)	pounds	173,120
Iron ore	29,366,654	(32,471,550)	long tons	36,708,318
Lead, white	118,079	(112,700)	short tons	11,807,900
Lead, red	13,800	(12,300)	short tons	1,759,500
Lead, orange mineral	800	(1,000)	short tons	124,000
Limestone and dolomite flux	8,788,000	(12,029,719)	long tons	3,515,200
Litharge	14,200	(12,400)	short tons	1,597,500
Phosphate rock	1,782,503	(1,570,228)	long tons	5,703,582
Pyrite	181,763	(199,387)	long tons	647,276
Sulphur	191,250	(39,310)	long tons	8,139,600
Zinc-lead	6,781	(4,500)	short tons	398,384
Zinc oxide	58,808	(64,034)	short tons	5,513,833
Zinc ore, exported	31,392	(39,418)	short tons	882,516
Total non-metallic				$629,295,184
Metallic.				
Aluminium	.....	(7,500,000)	pounds	.....
Antimony (existing only only as a constituent (20%) of hard lead)	4,633,036	(6,174,000)	pounds	$301,147
Bismuth (contents of ore mined, but not smelted in United States)	10,891	(no)	pounds	22,871
Copper	748,540,800	(689,045,796)	pounds	96,588,660
Gold	4,090,532	(3,560,000)	troy ounces	84,551,300
Iron, pig	16,563,956	(18,009,252)	long tons	219,474,679
Lead	313,553	(276,694)	short tons	27,021,997
Nickel	100,000	(114,200)	pounds	47,000
Platinum	120	(110)	troy ounces	2,340
Quicksilver	43,499	(37,866)	flasks of 75 lbs.	1,783,459
Silver	53,603,000	(54,300,000)	troy ounces	30,672,173
Zinc	176,849	(158,502)	short tons	18,038,598
Total metallic				$477,504,224

The same issue contained a great many special statistical articles by various authors on the various minerals, metals and markets during 1904.

RECENT METALLURGICAL PATENTS.

Smelting Ores in Blast Furnaces.

J. E. Johnson, Jr. (788,044, February 21) purposes to feed separate bodies of ore and fuel into the blast furnace, to smelt the charge and increase the amount of heat available for smelting, and especially for those reactions which require a high temperature, by supplying a blast containing an excess of oxygen and free from moisture. He passes the gases escaping from the furnace through the incoming body of ore and maintains them out of contact with the incoming fuel, the waste gases thus being high in carbon dioxide and low in carbon monoxide and nitrogen.

Treatment of Matte.

Two patents of O. S. Garretson (782,123 and 782,124, February 7) relate to a metallurgical operation which involves the treatment of matte by the converting or bessemerizing process in a blast furnace underneath a column of ore and flux or silicious material. When rich mattes are thus produced, the slag is correspondingly rich and carries off values which should be recovered in order to render the process economical. The inventor proposes to combine the stack furnace for the converting or bessemerizing process with a reverberatory furnace or forehearth which receives the slag from the converting furnace. He supplies sulphur-bearing material to the slag in the forehearth, while subjecting the slag in a shallow layer to heat applied from above the level of the slag.

He thereby produces low-grade matte which combines with metal or matte contained in the slag, a flow of this low-grade matter being maintained back to the bessemerizing or converting furnace.

In his second patent a settling well is provided which receives the slag from the slag-spout of the converting furnace. The matte which settles there is drawn off from time to time, while the slag is treated in the foreharth as before.

Hardening Iron.

F. L. Ramon (784,124, March 7) patents a process of hardening iron, by alloying it with 12 to 15 per cent aluminium. The iron and aluminium are brought to white heat in a crucible; a small quantity of sulphur is then placed on the metals and the contents "are reduced to the molten state by the application of additional heat, so as to be ready for pouring for the formation of the finished product. The said product may be made in an electric furnace or coke furnace, if preferred. By the use of an electric furnace the process will require about one minute; but by using a coke furnace, the same will require from two to three hours." The hardness of the product is increased with the use of additional parts of aluminium, the maximum hardness being obtained by the use of 12 per cent. "The said product weighs about one-half the weight of an equal quantity of steel and is tougher than the same."

Copper.

G. H. Waterbury (783,600, February 28) extracts copper from ore by the following wet method. The pulverized ore is placed in a tank containing a solution of sulphurous acid, to which is added a compartively small quantity of sulphuric acid, and air and steam are forced upward through the ore from the bottom of the tank. When the leaching operation is complete, the solution is drawn off into a precipitating tank containing particles of metal, as aluminium or steel, through which the solution freely circulates. Air and steam are then introduced at the bottom of the precipitating tank and caused to pass up through the copper solution, whereby the copper is precipitated or caused to settle on the bottom of the tank.

Agitating Device.

W. B. Devereux (781,406, January 31) patents an agitating device in which liquids with solid particles are first agitated, then settled and finally decanted. The invention consists, primarily, in combining with a tank a propeller or a propeller pump or other mechanical equivalent of the ordinary marine propeller rotating within the tank at such a depth that the solid particles, after settling, will not interfere with the starting of the propeller, with a series of radial diaphragms placed vertically within the tank, so constructed that the liquids with solid particles contained therein will pass freely through, around, and under them. The purpose of the diaphragms is to prevent rotation of the material in the tank when being agitated without subdividing the tank into separate or independent compartments.

Hascal A. Hogel and Herbert A. Hogel (781,520 and 781,521, January 31) patent a wet process in which the pulverized ore is subjected to the action of dissolving solution and the solvent action is intensified by "atomizing" the mixed mass under the influence of heat. The main point is the atomizing action by means of an aspirator of a certain construction.

Recovery of Volatile Acids from Solutions.

E. R. Hewitt (783,783, February 28) patents an apparatus for recovering volatile acids from solutions, for instance, sulphurous acid from an acid solution containing soluble salts, the method is especially applicable in the leaching of bones for the separation of the bone salts. The apparatus consists essentially of a tank with an open-ended pipe extending into the tank near its bottom, for the admission of live steam, and a curved condensing-chamber surrounding the open end of the pipe, whereby condensation of the steam is induced and the liquid in the receptacle is given a whirling motion. As soon as the liquid becomes hot enough, so that all the steam is not condensed in the body of liquid, the sulphurous acid is carried mechanically by the steam up into a condenser, where the acid separates in the form of gas which is forced upward into a pipe.

Miscellaneous.

T. McDonald (781,615, January 31) patents details of construction of the deflectors employed in connection with blast-furnace-charging devices to distribute the charge. This deflector is made vertically movable, so as to be raised into inoperative position for a part of the charges, and lowered into operative position for other parts of the charges.

T. McDonald (781,293, January 31) patents a cinder-ladle consisting of an outer ladle and an inner thimble, so supported that its bottom is not in contact with the ladle; the space between the bottoms of ladle and thimble is packed with a refractory material, the removable bottom is secured to the thimble by means not in contact with the outer ladle.

BOOK REVIEWS.

Jahrbuch der Elektrochemie und angewandten Physikalischen Chemie.—Berichte über die Fortschritte des Yahres 1903. Edited by Dr. Heinrich Danneel. Halle: Wilhelm Knapp; 930 pages. Price, 26.00 marks.

Larger, more imposing, and later than ever before, we welcome this permanent record of the yearly advances of electrochemistry. It must be said that electrochemistry is moving too rapidly for a reproduction of the work of 1903, at this period, to command the interest which it would have done if published during 1904. It may be put down as a truism, proved by experience, that unless an annual review is published within the limits of the year succeeding the one which it chronicles, that it loses at least one-fourth of its value, and one-half of the interest which it would otherwise command.

We note also the extension of the title from the simple "Jahrbuch der Electrochemie," corresponding to the extension of the title of the "Zeitschrift für Electrochemie," and the change of title of the "Deutsche Elektrochemische Gesellschaft." Here we believe that the interests of concentration of purpose and timely appearance have suffered, that the book loses part of its individuality, becomes unwieldy and necessarily late in appearing. The title is also meaningless; for applied physical chemistry, if there is, properly speaking, such a thing, would refer equally to all the practical operations of analytical chemistry, technical testing, manufacturing chemistry, and much of practical metallurgy. The title is too diffuse, and an attempt to live up to it, in the compass of one volume, necessarily results in imperfect success and incompleteness, not to speak of the resulting delay in preparation and publication.

The part of the book which is not electrochemical contains some most interesting information about the advances in physical chemistry and some of its applications, but does not compare in completeness or in satisfactory treatment with the electrochemical portion. The compilation of electrochemical books published in 1903 would be more satisfactory if it really represented all that were published, instead of merely those which were received by the editor. This list ought to be as complete as possible, in order to correspond with the main electrochemical part of the book.

The best that the reviewer can say of this volume is that it contains a fine abstract of all the advances in electrochemistry in 1903, and in this respect is quite equal in value to its invaluable predecessors. This is not too high praise, and should be sufficient incentive to every electrochemist who feads German to purchase the book.

Pneumatic Pyrometer.

One new type of pyrometer which has found much favor in iron and steel plants, especially for indicating and regulating the temperature of hot blast for blast furnaces, and also in connection with annealing and tempering furnaces, is the Uehling pneumatic pyrometer with Steinbart automatic recorder, which deserves a more detailed description on account of the very ingenious yet simple principle on which its operation is based, and which differs radically from the methods utilized in other types of pyrometers.

The Uehling pneumatic pyrometer is built by the Uehling & Decker Co., of Passaic, N. J. The principle of the instrument may best be explained with reference to Fig. 1. If two apertures, *A* and *B*, respectively form the inlet and outlet openings of a chamber *C*, and a uniform suction is created in the chamber *C'* by the aspirator *D*, the action will be as follows: Air will be drawn through the aperture *B* into the chamber *C'*, creating suction in chamber *C*, which in turn causes air from the atmosphere to flow in through the aperture *A*. The velocity with which the air enters through *A* depends on the suction in the chamber *C*, and the velocity at which it flows out through *B* depends upon the excess of suction in *C'* over that existing in the chamber *C*, that is, the effective suction in *C'*. The total suction remaining constant, the effective suction must decrease as the suction in *C* increases, hence the velocity at which air flows in through the aperture *A* increases, and the velocity at which air flows out through the aperture *B* decreases, until the same quantity of air enters at *A* as passes out at *B*. As soon as this occurs no further change of suction can take place in the chamber *C*. If the apertures *A* and *B* are of the same size, and the temperature of the air is the same while flowing through both, then equilibrium is established when the suction in *C* is one-half the suction in *C'*.

Air is very materially expended by heat. Therefore, the higher the temperature of the air the greater the volume, and the smaller will be the quantity of air drawn through a given aperture by the same suction. Now, if the air, as it passes through the aperture *A* is heated, but again cooled to a lower fixed temperature before it passes through the aperture *B*, less air will enter through the aperture *A* than is drawn out through the aperture *B*. Hence the suction in *C* must increase and the effective suction in *C'* must decrease, and in consequence the velocity of the air through *A* will increase and the velocity of the air through *B* will decrease, until the same quantity of air again flows through both apertures. Thus, every change of temperature in the air entering through the aperture *A* will cause a corresponding change of suction in the chamber *C*. If two manometer tubes *p* and *q*, Fig. 1, communicate respectively with the chambers *C* and *C'*, the column in tube *q* will indicate the constant suction in *C'* and the column in tube *p* will indicate the variable suction in the chamber *C*, which suction is a true measure of the temperature of the air entering through the aperature *A*.

It will, of course, be understood that to embody the above described principle in a practical pyrometer, certain conditions must be strictly fulfilled. Fig. 2 shows a diagrammatic disposition of all the parts combined in the complete instrument. The interior of the pipe *e, f, g, h, i* from aperture to aperture, together with the branches *q* and *s*, constitute the chamber *C* of Fig. 1. Its inlet from the atmosphere is through the opening *a* at the bottom of filter *I*, and its connection with chamber *C* is through the pipe *l*.

The aspirator *D* exhausts into the chamber *G*, keeping it at a constant temperature 212°. The steam and condensed water, together with the air drawn through the aperture, escape through the pipe *t* at atmospheric pressure. Opening the valve 6, steam enters the aspirator *D*, and sucks the air through the tube *m*, out of the chamber *C'* and produces a suction, which is kept constant by the regulator *H*, as shown by the manometer *p*. By keeping the water in *H* at a constant height, a perfectly uniform suction is maintined. With a constant suction in *C'* and cocks 2 and 4 open, air will enter at *a*, pass through the filter *I*, where it is purified through the connection *b* into the "fire tube." It flows forward in the annular space between the two tubes *c* and *f*; as soon as it reaches the platinum tube *d*, which protrudes from the cooler, it becomes heated and enters through the aperture *A* into the chamber *C*, at the temperature surrounding the exposed end of the fire tube, which is the temperature to be measured. After passing *A*, the air flows through the pipe *e, f, g, h*, into the coil *i*, where it assumes the temperature of 212°, at which it passes through aperture *B*, thence by the connection *l* into the chamber *C'*, from which it is drawn by the aspirator *D* through *m*, and discharged with exhaust steam and condensed water.

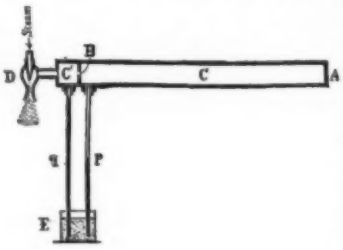


FIG. 1.—PRINCIPLE OF PNEUMATIC PYROMETER.

The branch pipes *s* and *q'* connect respectively with the recording gauge *L*, and the manometer *q*, which is placed in proximity to the temperature scale, as shown. This combination, therefore, fulfills all the conditions—viz., air is drawn through the instrument by a constant suction. It passes through aperture *B* at a constant temperature. Aperture *A* is so located that the air must enter at the temperature to be measured; hence, the indication of the manometer *q* will vary with the temperature at *A*, and the latter can be read off directly on the temperature scale placed beside the same.

The pyrometer is used in combination with the Steinbart recording gauge which has certain distinct advantages. There are no springs to get out of adjustment. The only point for friction is the contact of the pen against the paper. The temperature lines are printed on the paper just before passing under the pen, so that no error can occur by any movement of the paper up or down the feeding rollers. The gauge is driven

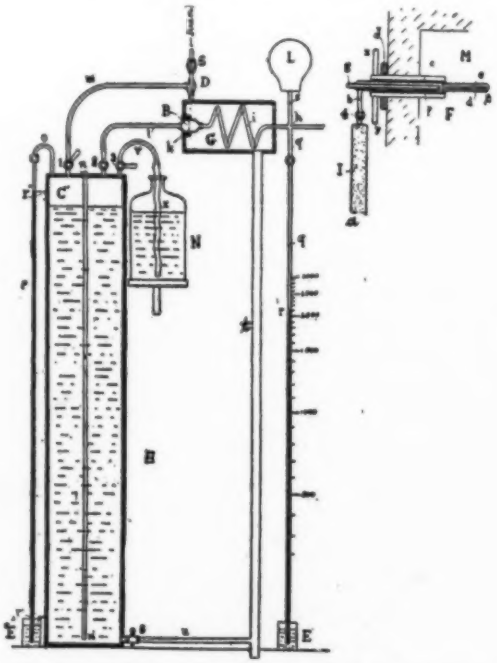


FIG. 2.—DIAGRAM OF PARTS OF PNEUMATIC PYROMETER.

by powerful clock-work actuated by two mainsprings, each capable of keeping the gauge going eight days. Seven hours of the record are always in view, and the whole can be seen by pulling it off the receiving roller. The working of the gauge is not sensitive to vibration; it will work quite well in the vicinity of moving machinery.

On account of these inherent advantages, the instrument has found much favor with practical furnace men, there being now more than 500 instruments in use in blast and annealing furnaces. Of these 30 or 40 are in England, 20 in Austria, 3 in Japan, 4 or 5 in France, and 8 or 10 in Germany, the balance being used in this country. The majority of the blast furnace managers prefer the double form of the instrument which indicates the temperatures of both the blast and gas.

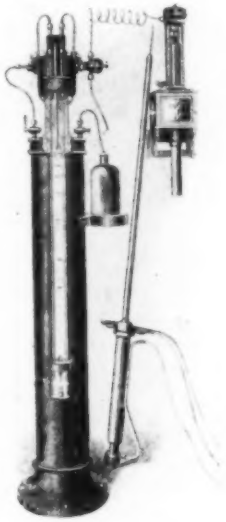


FIG. 3.—PNEUMATIC PYROMETER.

Fig. 3 is an illustration of the "simple pyrometer" with portable fire tube. The recording device is shown at the right side and is firmly connected with the main instrument on the left of the illustration. The portable fire tube is connected with the regulator by means of a rubber hose, so that the temperature may be measured at various points within a circle of about 30 meters.

Wheatstone Bridge Resistance Box.

The most compart and the most easily portable form of Wheatstone bridge is undoubtedly the plug contact form in which plug resistances are substituted for the stretched wire of the meter bridge. In the following we give a description of the "University of Texas Resistance Box," as built by the L. E. Knott Apparatus Company, Boston, Mass. It was designed with a view of obtaining the highest degree of precision of which the plug form of Wheatstone bridge is capable. The accuracy of a resistance box in practical use depends upon a number of conditions of which the mere accuracy of adjustment is but one. In the University of Texas Resistance Box, the object was to provide permanency of calibration in every detail.

The coils of this box are:

Rheostat arm,			
1	10	100	1000 ohms
2	20	200	2000 "
3	30	300	3000 "
4	40	400	4000 "
Ratio arms,			
1	10	100	1000 "
	10	100	1000 "
Total.....			13,331 "

The one-ohm ratio coil can be thrown into either arm desired by means of one of the traveling plugs supplied with the box. An additional coil of 10,000 ohms is provided if desired. This coil can be thrown into either the rheostat arm or the nearer ratio arm, or can be cut out altogether by means of the ordinary bridge plugs; or by the traveling plugs it can be thrown into the other ratio arm. It is therefore available for a variety of purposes.

The theoretical range of the bridge would therefore be from

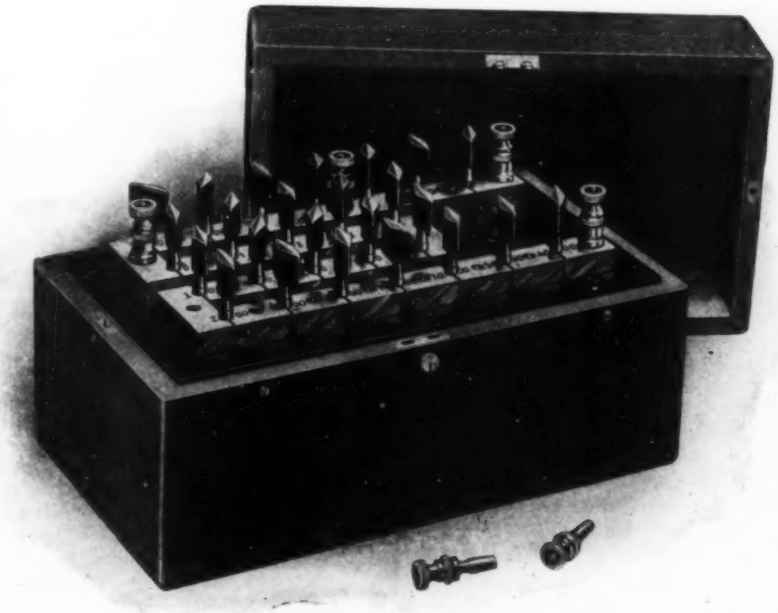
1-10,000th ohm to 111,100,000 ohms;

or without the additional 10,000 ohm coil,

1-1,000th ohm to 11,110,000 ohms.

The actual working range of any plug box is, of course, far inside even of the narrower of the above limits; being determined by plug contact resistances on the one hand, and insulation provisions on the other.

The coils of the rheostat arm are adjusted to 1-10th per cent, and those of the ratio arm to 1-20th per cent. The coils are made of manganin wire, which has a temperature coefficient practically zero over the limits of laboratory variations. Moreover, these coils are wound anti-inductively on highly insulated metal bobbins of special construction and open at both ends, which gives thorough ventilation from inside and outside of coils. In this way the heat which is developed in the coils is easily carried off.



UNIVERSITY OF TEXAS RESISTANCE BOX.

The coils are mounted in a new manner which makes each independent of the others, so that in case of accident it can be removed, readjusted and replaced without disturbing the rest. In this method of mounting, each coil is elevated on short ebonite pillars, giving enormous insulation resistance at the interior of the box, which is not so easily accessible for cleaning.

The arrangement of the connection blocks was adopted after careful consideration of the dial patterns and other group arrangements as being simpler and capable of more perfect cleaning and insulation. It is the modern double-bent arrangement, which brings the X terminals at the right-hand end of the

bridge, so that the box can be conveniently joined to another box, such as a sub-divided ohm, or to a slide wire. This arrangement, moreover, gives connections to the "unknown" which are of equal length and resistance. It also results in the most symmetrical and convenient grouping of the coil values.

The shanks of the bridge plugs are of special shape, which prevents loss of fit by wear of the plugs or holes—a small but very important detail. The shanks are of large diameter and, being most carefully fitted, give contacts of very low resistance. The taper of the plugs is carefully chosen, so as to prevent wedging the blocks apart and yet to allow the insertion of the plugs without undue forcing.

An important new feature of the box is the addition of a heavy metal girder frame securely screwed to the substantial ebonite top. This keeps the ebonite absolutely rigid and allows of perfect alignment of the brass blocks, and of fitting the plugs to their holes perfectly and permanently. Between the ebonite and the sides of the box is placed a felt gasket, making the box dust-tight and practically air-tight.

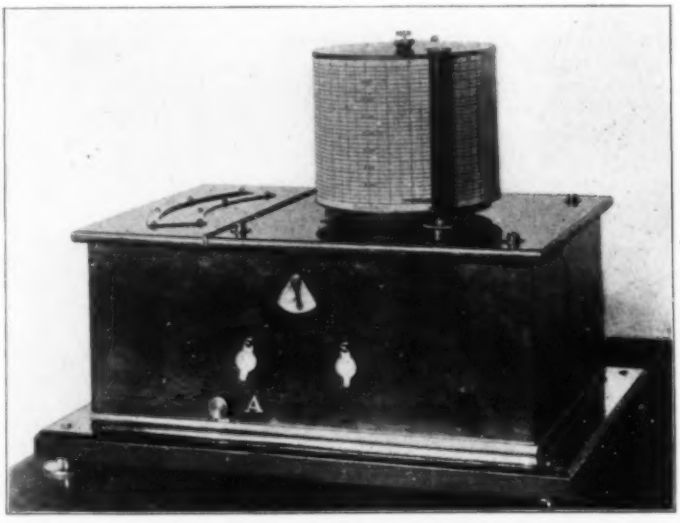
To keep the high initial insulation resistance permanent, reasonable care is, of course, necessary in any resistance box; to make this easy, special facilities for cleaning are provided, and to this end also acts a wooden box cover which is always placed on the box when it is not in service.

Recording Electric Pyrometer.

Many metallurgical and chemical processes, in order to be as efficient as possible, depend on exactly maintaining the required temperature. Maintenance or regulation of temperatures requires, of course, the possibility of measuring them exactly. As long as these temperatures are relatively low, the ordinary thermometer is sufficient. But the problem is of no less importance for the reactions at higher temperatures, and an enormous amount of ingenuity, hard work and good money has been spent in recent years for developing exact pyrometers.

The electric pyrometer using a thermo-couple has been known in principle for a very long while, since Pouillet made heat determinations on this principle as early as 1836. The electric pyrometer was later brought into prominence by Prof. Henry LeChatelier, and was introduced in a more practical form by Prof. Robert Austin, who provided it with a photographic recording adjustment. This latter point, however, has for a long while been a distinct disadvantage in the use of the electric pyrometer for many metallurgical purposes. When the record was taken by photography it had to be developed and fixed before it could be seen, so that the record was practically twenty-four hours behind the time when it was taken. Now, this is, of course, undesirable, since, after twenty-four hours have elapsed it is too late to make changes in the operation of the furnace which could have been easily made if the attendant had the record always in front of him, showing him the temperature at every instant.

In view of these considerations a new pyrometer made by Messrs. Queen & Co., of Philadelphia, is very interesting, since the former disadvantages of the electric type of pyrometer have been successfully overcome in this instrument. It makes a continuous record of temperatures up to 3000° F. It consists of a sensitive D'Arsonval galvanometer, the recording mechanism and the thermoelectric couple. The thermoelectric couple consists of the regular platinum and platinum rhodium wires 3 feet long, although a couple of any length will be furnished if desired. The D'Arsonval galvanometer measures the electric current produced by the heating of the junction of the thermoelectric couple. It was always perfectly understood that the electric pyrometer is a most delicate and accurate instrument, but the objection was sometimes raised that this delicacy renders the instrument occasionally difficult to use, for instance, for the reason that it is sensitive to vibrations. The galvanometer used in the Queen instrument is specially designed for its particular work. It is extremely sensitive, absolutely dead beat and so constructed that vibrations will not effect it.



RECORDING ELECTRIC PYROMETER.

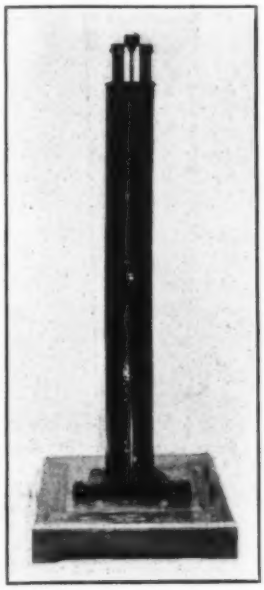
The recorder is so arranged that the pen traces on the chart the exact movements of the galvanometer pointer. The cylinder on which the chart is fastened revolves once in twenty-four hours. In the clock that drives the cylinder an electric contact is made every minute. This intermittent current, by passing through an electromagnet, clamps the pointer of the galvanometer, which in turn operates a relay and causes the pen to move either up or down, until it coincides with the reading of the galvanometer, when the pointer is released and swings entirely free until the next contact is made, thus the recording device in no way impairs the accuracy or sensibility of the pyrometer. It will thus be seen that in the Queen recording pyrometer all the well-known advantages inherent in the electric pyrometer have been reserved, while the former disadvantages of this type of instrument have been successfully overcome.

Standpipe Rheostats.

A series of rheostats has recently been designd by Messrs. Queen & Co., of Philadelphia, Pa., consisting of a continuous length of wire wound spirally on an iron standpipe, about 6 inches in diameter, of a length depending on the total resistance and the current carrying capacity required.

The wire is insulated from the iron pipe to withstand an alternating potential of 1000 volts. The pipe is closed at the bottom, and by means of a small pipe, on either side of the base, a stream of water can be introduced for cooling purposes. If the rheostats are not used at their maximum capacity, water is not necessary; when only used at their maximum capacity for a few minutes at a time, the pipe filled with water alone is sufficient; running water is recommended only when the rheostats will have hard continuous service at their maximum rated capacity. The standpipe is mounted on a hardwood

base, provided with castors, so as to be moved about the laboratory easily. Mounted parallel to the upright pipe is an angle-iron guide, which carries a slider, to which are attached phosphor bronze springs, which, in turn, make contact against the wire on the pipe. All the parts are insulated except a rod in the center of the angle-iron guides, which is connected with the phosphor bronze springs by means of a split brass sleeve in the wooden handle. This gives a very smooth, even motion, and positive contact is made on every convolution of the wire, which allows adjustments of the current by almost infinitesimal steps.



STANDPIPE RHEOSTAT.

These rheostats are especially suitable for laboratory work in electrolysis, where it is necessary to secure fine graduations of both current and potential. To secure a potential regulation the rheostat is supplied with two sliders, the terminals of the main conductor are connected with the main source of supply, and the wires to the apparatus are connected to the two sliders. By putting the sliders close together, the difference of potential is very small, whereas, by sliding them farther apart, it can be increased up to any desired point.

Centripact Screen.

In our last issue we described in an article on the concentrating plant of the Green Mining & Milling Co., of Silverton, Colo., a novel and interesting screening device, the Centripact screen of the Traylor Engineering Co., of New York City. By a mistake of the printer, however, Fig. 5, on page 125, was reversed. We therefore give again the two illustrations of the screen. Fig. 1 shows the double-unit Centripact screen for dry work. As was explained in our last issue, the screen is approximately saucer-shaped, and is mounted on a vertical shaft which revolves, while simultaneously successive vertical impacts are given to the shaft and screen, so that the particles on the screen are subjected to two forces—the centrifugal force tending to drive the particles towards the periphery of the screen, and an upward impact tending to make them leap from the surface.

FIG. 1.—DOUBLE-UNIT CENTRIPACT SCREEN FOR DRY WORK.

Fig. 2 is a section of a centripact screen unit for wet work. The most essential feature of this type is the provision of a sheet-steel pan conforming in shape to the screen and supported beneath the screen cloth about ¾ of an inch from the same. The water passing through the screen is caught in the

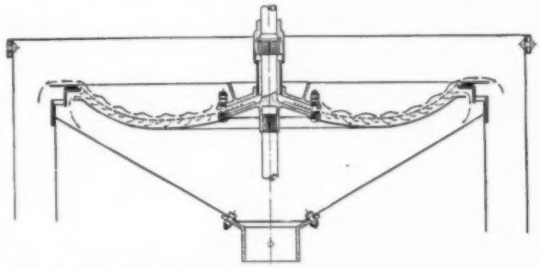


FIG. 2.—SECTION OF CENTRIPACT SCREEN UNIT FOR WET WORK.

pan and flows outward over it. Owing to the action of centrifugal force, the surface of the water assumes a form corresponding to that of the screen and pan. There is thus a layer of water immediately under the screen, and with each impact, the water is projected upward against the screen, serving to clean the meshes and at the same time thoroughly irrigating the material to be screened.

Industrial Notes.

The J. T. Baker Chemical Co., in Easton, Pa., are producing a full line of high- grade chemicals for technical and analytical work. Mr. J. T. Baker, formerly of the Baker & Adamson Chemical Co., is president of the new concern. The plant is in Phillipsburg, N. Y., and comprises several new and commodious buildings. A fully equipped laboratory is maintained in a separate building in which the offices of the concern are located.

We have received from *The Fossil Flour Co.*, of 229 Pearl Street, New York, a folder on the use of fossil flour in rubber compounds, as a "rubber drug;" when added to any zinc or whiting compound, it is stated to increase the strength, increase the bulk and reduce the cost. Infusorial earth may also be used to good advantage as a heat insulating material in electric furnaces, as pointed out on page 55 of our February issue, and on page 139 of the present issue.

A clearly written and well illustrated pamphlet of the *Westinghouse Electric & Mfg. Co.* (Folder No. 4032) deals with wattmeters and how to read them. Circular 1099 deals with bipolar motors, type R, for direct-current circuits, circular 1097 with type K motors, direct current, series wound for crane, hoisting and similar service.

The *De La Vergne Machine Co.*, of New York City, has recently issued two nicely illustrated folders. One deals with economical gas and oil engines: "The gas plant of small capacity differs widely from the small steam plant, in that it enjoys an efficiency equal to the large gas plant, and not only saves the losses due to electrical transmission, but also attains an efficiency far superior to the steam-driven prime movers of the large central stations supplying the electricity. The development of small power high efficiency gas engines designed to utilize natural or producer gas is bound to react eventually, causing an increase in the number of small isolated plants at the expense of large installations." Several Koerting gas engines and Hornsby-Akroyd oil engines, built by the De La Vergne Machine Co., are illustrated. The other folder refers to refrigerating and ice-making machinery.

Electricity in Copper Mines.—The Mammoth Copper Mining Co., of Salt Lake City, has just placed an order with the Westinghouse Electric & Manufacturing Co. for a large addition to their electrical equipment. A total of nearly 800 horse-power in induction motors of different sizes, together

with a motor-generator set, transformers, switchboard and three mining locomotives, is included in the order. The motors, two of which have a rated capacity of 200 horse-power each, and five of 50 horse-power each, will operate on three-phase 2000-volt circuits. This order is merely indicative of the progress being made in the use of electric power in mining operations.

Flange Joint Gaskets.—The history of flange joint packing has been, of necessity, closely identified with and influenced by that of all steam service. As the temperatures and pressures have increased, the earlier forms of flange gaskets have become less and less efficient, necessitating continual improvements to meet the severity of modern requirements. The advantages of asbestos for this purpose are described in a pamphlet "Flange Joint Gaskets," issued by H. W. Johns-Manville Co., 100 William Street, New York.

The *Charles E. Sholes Co.* announces the establishment and equipment of industrial laboratories for chemical research and analysis at 164 Front Street, New York, with Dr. J. E. Teeple, formerly instructor at Cornell, as director.

Digest of U. S. Patents.

Prior to July, 1902.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Electric Smelting and Reduction Processes.

(*Continued from page 128.*)

430,457. June 17, 1890. Thomas L. Willson, Brooklyn. New York.

Protects carbon electrodes and vessels by the introduction of a reducing gas, such as hydrogen or illuminating gas. The furnace shown is a carbon crucible resting on a carbon plate having a terminal, surrounded by a wall and covered by a slab having a vent hole. A depending carbon pencil passes frictionally through an opening in the slab. The carbon pencil may be tubular, for the introduction of gas, or a separate pipe of refractory material may be employed. The tubular electrode may be built up of four plates arranged in a rectangle, luted at the joints with a paste of carbon dust and molasses and held together by metal clamps and carbon dowel-pins. The conductor consists of spaced copper wires leading to a bar terminal. Produces boron, silicon, calcium, chromium, titanium, iron, steel, etc. For the production of aluminium bronze, two parts of broken copper or copper wire are placed in the bottom of the crucible and covered by one part of corundum. The material is smelted by an arc, from four to five pounds of material requiring 20 amperes at 50 volts. The direct current may be passed in either direction, preferably from the crucible cathode to the depending electrode, to prevent rapid consumption of the latter. The copper is melted and vaporized and the copper vapors circulate within the crucibles, condensing against the relatively cool sides and streaming down through the corundum. The operation requires from 15 minutes to 2 hours, the deoxidizing gas being supplied throughout. By partially melting down the fireclay cover, an aluminium silicon bronze is produced. By smelting iron oxide containing corundum, aluminium steel is produced. Aluminium bronze may be made from a charge of alumina and copper sulphide, *e. g.*, redruthite, the eliminated oxygen and sulphur being combined with a hydrocarbon gas.

486,575. November 22, 1892. Thomas L. Willson, Leaksville, N. C.

Production of aluminium bronze by electrical reduction. Alumina, in a state of fine sub-division and preferably in a porous condition, is thoroughly impregnated with a hydrocarbon liquid such as coal-tar, heavy hydrocarbon oil or paraffin. The hydrocarbon is heated to its boiling point and the alumina is added little by little with constant stirring, until the hydrocarbon is completely absorbed. When precipitated alumina is used the proportions are about 22 pounds of the alumina to each gallon of tar. Reduction is effected either by heating the product as above obtained in an incandescent electric furnace, or in an arc furnace of the Siemens type, the mixed alumina and tar being mingled with broken carbon in the former case. In either case granulated copper is supplied in sufficient proportion to absorb the reduced aluminium with the production of aluminium bronze. The statement is made that sodium chloride may replace the hydrocarbon as a reducing agent, chlorine escaping and the reduction of the alumina being effected by the sodium.

491,394. February 7, 1893. T. L. Willson, Brooklyn, N. Y.

A furnace for the reduction of refractory oxides comprises a carbon crucible resting upon a carbon plate and connected as the anode of an electric circuit, the cathode being a carbon pencil projecting into the crucible through a luted cover. For the production of aluminium bronze the charge consists of finely divided and preferably precipitated alumina, and finely powdered carbon, the oxide and the reducing agent being either mixed or fed alternately into the furnace. A suitable quantity of copper is added for the purpose of taking up the molten aluminium as it is liberated. The temperature of the furnace being above the vaporizing point of copper, this metal is constantly distilled and condenses in the upper portion of the furnace, the liquid copper flowing back through the charge and extracting from it the particles of aluminium. Aluminium steel is said to be produced by the simultaneous reduction of iron ore and corundum or by reducing an aluminium compound in the presence of fragments of iron. The process is stated to be applicable to the reduction and alloying of other highly refractory elements, such as boron, silicon, calcium, chromium, titanium, etc.

492,377. February 21, 1893. T. L. Willson, Leaksville, N. C.

The furnace comprises a carbon or graphite crucible, into which depends a carbon pencil, the crucible and pencil constituting opposite terminals of the electric circuit. Carbon plates resting upon the masonry walls enclosing the crucible serve to close the furnace. It is stated that when aluminium bronze is prepared by the reduction of alumina in presence of copper, and in all other reactions in which a molten mass is present in the furnace, there are violent fluctuations in the resistance of the furnace, which occur by reason of the ebullition of the fused bath. The present invention consists in suppressing this ebullition by the use of carbon in sufficient proportions to prevent the formation of a fused bath; under these conditions the operation proceeds quietly and with relatively high economy. The carbon may be added in the form of a hydrocarbon, the mixture being subsequently coked, or in the form of finely divided coke or other material. For the reduction of alumina it is found that carbon equal to 15 per cent by weight of the mixture is sufficient to suppress the fused bath and accomplish the above results. The aluminium may be recovered by introducing an alloying base metal, as copper, into the furnace, or it may be recovered as aluminium carbide in presence of the excess of carbon. The furnace may be modified as desired, for instance, by striking the arc between two carbon pencils in close proximity to the charge, but incandescent furnaces are excluded. It is stated that the process is applicable to the reduction of many metals other than aluminium, and the following statement constitutes the first patent reference to the electric furnace production of calcium carbide:

"For example, I propose to apply it (the process) for the treatment of refractory compounds or ores of metals, not necessarily for the production of the metals themselves, but for the production of other compounds thereof. For example, I have already employed it for reducing calcium oxide and producing calcium carbide."